

Abram S. Hewitt

# TRANSACTIONS

OF THE

## AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXXIV.

---

CONTAINING THE PAPERS AND DISCUSSIONS OF 1903.

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NEW YORK CITY:  
PUBLISHED BY THE INSTITUTE,  
AT THE OFFICE OF THE SECRETARY.

1904.



## PREFACE.

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THIS book contains, in addition to the Proceedings, Papers and Discussions of 1903, some communications received in 1904, which are here published in order to put them in the same volume with the papers to which they refer.

The labor required for the publication in 1902 of Vol. XXXII. (the "Mexican Volume"), in addition to Vol. XXXI., had involved, as a consequence, a regrettable delay in the issue of Vol. XXXIII., which did not take place until March, 1904. A strenuous effort has been made to recover the ground thus lost; and I am glad to say that, through the efficient aid of Dr. Joseph Struthers, the Assistant Editor of the *Transactions*, this effort has been measurably successful.

R. W. RAYMOND.



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## OFFICERS.\*

For year ending February, 1904.

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ALBERT R. LEDOUX, New York City.

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S. F. EMMONS..... Washington, D. C.  
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J. HENRY LEE..... Baltimore, Md.  
(Term expires February, 1904.)

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PHILIP W. MOEN..... Worcester, Mass.  
JAMES F. KEMP..... New York, N. Y.  
(Term expires February, 1905.)

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(Term expires February, 1904.)

E. W. PARKER..... Washington, D. C.  
JAMES W. NEILL..... Salt Lake City, Utah.  
M. D. VALENTINE..... Woodbridge, N. J.  
(Term expires February, 1905.)

HEINRICH RIES..... Ithaca, N. Y.  
B. B. LAWRENCE..... New York, N. Y.  
F. KLEPETKO..... New York, N. Y.  
(Term expires February, 1906.)

### Secretary.

R. W. RAYMOND,  
99 John St., New York City.

### Treasurer.

FRANK LYMAN,†  
New York City.

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\* The following officers were elected at the Annual Meeting, February, 1904.

*President*, James Gayley, N. Y. City; *Vice-Presidents* (to serve two years), Julian Kennedy, Pittsburg, Pa.; C. D. Walcott, Washington, D. C.; George W. Maynard, N. Y. City. *Managers* (to serve three years), F. L. Grammer, Baltimore, Md.; Joseph Hartshorne, Pottstown, Pa.; Charles H. Snow, N. Y. City.

† By appointment of the Council, to fill the vacancy caused by the death of Mr. Theodore D. Rand, April 24, 1903.

## PAST OFFICERS.

## PRESIDENTS.

*DAVID THOMAS.....	1871
R. W. RAYMOND.....	1872
R. W. RAYMOND.....	1873
R. W. RAYMOND.....	1874
*A. L. HOLLEY.....	1875
*ABRAM S. HEWITT.....	1876
*T. STERRY HUNT.....	1877
*ECKLEY B. COXE.....	1878
*ECKLEY B. COXE.....	1879
*WILLIAM P. SHINN.....	1880
WILLIAM METCALF.....	1881
*RICHARD P. ROTHWELL.....	1882
ROBERT W. HUNT.....	1883
JAMES C. BAYLES.....	1884
JAMES C. BAYLES.....	1885
ROBERT H. RICHARDS.....	1886
*THOMAS EGLESTON.....	1887
WILLIAM B. POTTER.....	1888
RICHARD PEARCE.....	1889
*ABRAM S. HEWITT.....	1890
JOHN BIRKINBINE.....	1891
JOHN BIRKINBINE.....	1892
H. M. HOWE.....	1893
JOHN FRITZ.....	1894
*J. D. WEEKS.....	1895
E. G. SPILSBURY.....	1896
THOMAS M. DROWN.....	1897
C. KIRCHHOFF.....	1898
JAMES DOUGLAS.....	1899
JAMES DOUGLAS.....	1900
E. E. OLCOTT.....	1901
E. E. OLCOTT.....	1902
ALBERT R. LEDOUX.....	1903
JAMES GAYLEY.....	1904

## SECRETARIES.

*MARTIN CORYELL.....	1871-1872
THOMAS M. DROWN.....	1873-1884
R. W. RAYMOND.....	1884 —

## TREASURERS.

J. PRYOR WILLIAMSON.....	1871-1872
*THEODORE D. RAND.....	1872-1903
FRANK LYMAN.....	1903 —

\* Deceased.

## HONORARY MEMBERS.

PROF. RICHARD AKERMAN.....	Stockholm, Sweden.
SIR LOWTHIAN BELL.....	Middlesborough, England.
DR. THOMAS M. DROWN.....	South Bethlehem, Pa.
PROF. HATON DE LA GOUPILLIÈRE.....	Paris, France.
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PROF. DIMITRY CONSTANTIN TSCHERNOFF.....	St. Petersburg, Russia.
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III.	Troy, N. Y.	November, 1871	i.	13
IV.	Philadelphia, Pa.	February, 1872	i.	17
V.	New York, N. Y.*	May, 1872	i.	20
VI.	Pittsburg, Pa.	October, 1872	i.	25
VII.	Boston, Mass.	February, 1873	i.	28
VIII.	Philadelphia, Pa.*	May, 1873	ii.	3
IX.	Easton, Pa.	October, 1873	ii.	7
X.	New York, N. Y.	February, 1874	ii.	11
XI.	St. Louis, Mo.*	May, 1874	iii.	3
XII.	Hazleton, Pa.	October, 1874	iii.	8
XIII.	New Haven, Conn.	February, 1875	iii.	15
XIV.	Dover, N. J.*	May, 1875	iv.	3
XV.	Cleveland, O.	October, 1875	iv.	9
XVI.	Washington, D. C.	February, 1876	iv.	18
XVII.	Philadelphia, Pa.†	June, 1876	v.	3
XVIII.	Philadelphia, Pa.	October, 1876	v.	19
XIX.	New York, N. Y.	February, 1877	v.	27
XX.	Wilkes-Barre, Pa.*	May, 1877	vi.	3
XXI.	Amenia, N. Y.	October, 1877	vi.	10
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XXVII.	Montreal, Canada	September, 1879	viii.	121
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XXIX.	Lake Superior, Mich.	August, 1880	ix.	1
XXX.	Philadelphia, Pa.*	February, 1881	ix.	275
XXXI.	Staunton, Va.	May, 1881	x.	1
XXXII.	Harrisburg, Pa.	October, 1881	x.	119
XXXIII.	Washington, D. C.*	February, 1882	x.	225
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\* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

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				and xxxv.
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\* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida.

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## PUBLICATIONS.

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THE publications of the Institute comprise :

### TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty ; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers :

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*The Genesis of Ore-Deposits*, comprising the famous treatise of the late Professor Franz Posepny, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Posepny Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Vol-

ume XXX. and the remainder appeared in Volume XXXI.) ; also a complete bibliography of the Institute papers and discussions on this subject from 1871 to the present time.

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*List of Members, Rules, etc.*, paper, . . . . . 50

#### PAMPHLETS.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed, without charge, to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price, as follows :

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57 to 72 ".....	0 35	3 00	5 00
73 to 88 ".....	0 40	3 25	5 25
89 to 104 ".....	0 45	3 50	6 00
105 to 120 ".....	0 50	3 75	6 25

Papers with folders and inserted plates subject to special price.

### AUTHORS' EDITIONS OF PAMPHLETS.

Extra copies, when ordered *before* the printing of the pamphlet edition distributed to the members of the Institute, will be furnished to authors, under Rule VII., at special rates, which will be stated on application to the Secretary.

All communications and remittances should be addressed to R. W. Raymond, Secretary, 99 John St., or P. O. Box 223, New York City.

# RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881,  
1887, 1890, 1896, 1903 AND 1904.

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## I.

### OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

## II.

### MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote, and members or associates whose post-office address shall be outside of the United States, Canada and Mexico shall not be entitled to vote by mail, except upon proposed amendments to the Rules.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

### III.

#### DUES.

The dues of members and associates shall be ten dollars each per annum, payable in advance on the first day of each calendar year; and persons elected at any time during any calendar year shall pay the dues of that year upon election, and shall thereupon be entitled to the publications of the year, as if the said payment had been made at the beginning thereof. And, in addition to the above dues, the Council may at any time prescribe an initiation fee of ten dollars for all candidates thereafter proposed.\* Honorary members shall not be liable to dues. Any member or associate not in arrears may become, by the payment of one hundred and fifty dollars at one time, a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

\* NOTE.—This provision became operative April 18, 1904.

### IV.

#### OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary, and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed;

*Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum ; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

The Council may at any time appoint a trustee or trustees, to hold property, real or personal, for the use and benefit of the Institute, upon such terms and conditions as the Council may determine.

## V.

### ELECTIONS.

The annual election shall be conducted as follows : Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting ; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members) a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office ; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting ; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer ; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

## VI.

### MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Other meetings shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance.

Every question, which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting ; but the latter shall not take part in the proceedings without the consent of the meeting.

## VII.

## PAPERS AND PUBLICATIONS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute shall be printed in the *Transactions*. Intimation, when practical, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers. The published papers and volumes of *Transactions* shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the Rules.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade; nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute, as provided in these Rules; *Provided*, however, that committees may be appointed by the Council or the Institute to make investigations and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports.

## VIII.

## AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided*, also, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as Scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so

adopted ; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these Rules from and after the publication of said announcement by the Secretary.

## IX.

### SPECIAL AUTHORITY GIVEN TO THE COUNCIL.

The Council is hereby authorized to cause to be created under the statutes of the State of New York, a corporation to hold and administer, for the use and benefit of the Institute, such real or personal property as the Council may, from time to time, transfer to it, such transfer being hereby authorized ; said corporation not to be subject to control by the members and associates of the Institute, except through the Council, and in such way as may be permitted by the articles of its incorporation, approved by the Council. *Provided*, That the regular business of the Institute, namely, the holding of meetings, and the preparation and distribution of publications, shall remain completely in the control of the Council ; and that the funds of the Institute, derived from the fees and dues of members and associates or the sale of publications, shall not be transferred to, or held, or controlled by, such corporation, except so far as the Council of the Institute may appropriate money for the necessary expenses of the same—the purpose of this provision being, that the regular business of the Institute, as above defined, shall remain in the hands of the Council, without interference on the part of the said corporation.

And the Council is hereby authorized to give its assent and co-operation to the generous offer of Andrew Carnegie, a member of this Institute, to provide a building in the City of New York for the use of American Engineering Societies ; and in case a corporation should be formed to own and administer such building, the Council is authorized to name representatives for the Institute in the managing board of said corporation.





Proceedings of the Eighty-Fourth (Thirty-Third Annual)  
Meeting, Albany, New York, February, 1903.

ALBANY COMMITTEES.

*Executive Committee.*—General John H. Patterson, *Chairman*; William B. Jones, *Secretary*; Louis S. Treadwell, *Assistant Secretary*; A. Bleecker Banks, W. Howard Brown, Verplanck Colvin, Wm. Burgess Elmendorf, George P. Hilton, Edgar C. Leonard, Donald McDonald, F. J. H. Merrill, John DeWitt Peltz, John Boyd Thacher, Enrique Touceda, E. B. Toedt, Willis G. Tucker, M.D.

*Chairmen of Special Committees.*—General John H. Patterson (Reception); F. J. H. Merrill (Banquet); Wm. Burgess Elmendorf (Transportation); W. Howard Brown (Finance).

*General Committee.*—Verplanck Colvin, *Chairman*; William B. Jones, *Secretary*; Louis S. Treadwell, *Assistant Secretary*; C. E. Argersinger, Benjamin W. Arnold, Danforth E. Ainsworth, Rt. Rev. T. M. A. Burke, Rev. W. W. Battershall, Hon. A. Bleecker Banks, Edward Bowditch, J. Warner Bott, James P. Boyd, M.D., A. S. Brandow, Rev. E. H. Brown, William Barnes, Jr., William H. Brainerd, W. Howard Brown, Lewis Boss, Joel W. Burdick, A. E. Brainard, William Barnes, George D. Babbitt, Alpheus T. Bulkeley, J. M. Bailey, Anthony N. Brady, Hon. Alden Chester, James Fenimore Cooper, Verplanck Colvin, John M. Clarke, Hon. E. Countryman, Lewis E. Carr, Ledyard Cogswell, Rt. Rev. Wm. C. Doane, Curtis N. Douglas, Edward A. Durant, Philander Deming, Melvil Dewey, Wm. B. Elmendorf, James C. Farrell, J. Newton Fiero, A. W. Fuller, Hon. Charles H. Gaus, George E. Gorham, M.D., Hon. Clifford D. Gregory, Charles Gibson, Hon. Martin H. Glynn, William H. Griffith, Edward A. Groesbeck, Hon. D. Cady Herrick, Henry Hun, M.D., E. F. Hackett, Andrew Hamilton, Albert E. Hoyt, James Hendrick, Ernest Hoffman, Hugh Hastings, F. A. Harrington, Marcus T. Hun, Frederick Harris, George P. Hilton, John S. Hoy, Hon. David B. Hill, William B. Jones, Peter Kinnear, Gen. Rufus H. King, James B. Lyon, Hon. William L. Learned, Isaac D. F. Lansing, Rev. Archibald L. Love, Gerrit Y. Lansing, J. Townsend Lansing, Gardner C. Leonard, Edgar C. Leonard, William Lansing, C. W. Mead, Rev. David O. Mears, W. E. Milbank, M.D., H. E. Mereness, M.D., C. S. Merrill, M.D., Selden E. Marvin, Frederick G. Mather, F. J. H. Merrill, Henry T. Martin, W. G. MacDonald, Donald McDonald, M.D., James F. McElroy, James McCredie, Col. John V. McHarg, Hon. J. B. McEwan, Edward N. McKinney, John A. McCarty, John E. McElroy, Hon. J. T. McDonough, John W. McNamara, Elijah W. Murphey, Charles Newman, John T. Norton, John L. Newman, Robert Olcott, Charles G. Ogden, Dudley Olcott, Hon. Alton B. Parker, Rev. William Prall, Gen. J. H. Patterson, Gen. F. Phisterer, Col. Augustus Pruyn, Charles L. Pruyn, Charles H. Peck, Erastus D. Palmer, Walter Launt Palmer, John DeWitt Peltz, Foster Pruyn, Amasa J. Parker, John T. Perry, John D. Parsons, Robert C. Pruyn, John Pumpelly, Rev. C. A. Richmond, Henry Russell, Cuyler Reynolds, Marcus T. Reynolds, S. W. Rosendale, Charles H. Ramsey, William G. Rice, Rev. W. L. Robbins, Oscar D. Robinson, John W. Robe, William P. Rudd, Rev. E. G. Sel-

den, Rabbi M. Schlesinger, Rev. J. W. Sylvester, Hon. G. N. Southwick, Charles H. Sabin, Jared W. Scudder, Charles R. Skinner, Grange Sard, H. King Sturdee, W. O. Stillman, M.D., Hon. J. B. Thacher, George H. Thacher, W. G. Tucker, M.D., Col. G. C. Treadwell, Col. Richard H. Talcot, Charles H. Turner, E. B. Toedt, Samuel B. Townner, Enrique Touceda, Charles Tracey, Frederick Tillinghast, Gilbert M. Tucker, Luther H. Tucker, Jr., Hon. T. J. VanAlstyne, A. Vander Veer, M.D., H. Van Rensselaer, M.D., Maurice E. Viele, J. L. Van Valkenburgh, W. B. Van Rensselaer, G. A. Van Allen, Samuel B. Ward, M.D., Albert J. Wing, William M. Whitney, John E. Walker, William J. Walker, Rev. W. F. Whitaker, H. P. Whitlock, Henry P. Warren, Frederick E. Wadhams, James B. Wasson, Horace G. Young.

### SCHENECTADY COMMITTEE.

Citizens' Committee at Schenectady, N. Y., for entertaining delegates to the American Institute of Mining Engineers, Wednesday, February 18, 1903:

*For the General Electric Company.*—J. R. McKee, *Chairman*; F. O. Blackwell, H. W. Darling, W. L. R. Emmet, G. E. Emmons, S. M. Hamill, J. R. Lovejoy, M. A. Oudin, Hinsdill Parsons, E. W. Rice, A. L. Rohrer, C. P. Steinmetz, M. F. Westover.

*For the American Locomotive Company.*—H. C. Hecquembourg, James McNaughton, A. J. Pitkin, W. L. Reid, Matthew Taylor, A. W. White.

*For Union University.*—Prof. R. S. Curtis, A. V. V. Raymond, LL.D., Dean B. H. Ripton, Prof. J. H. Stoler, Prof. J. R. Truax, Prof. T. W. Wright.

*For the City of Schenectady.*—J. H. Callanan, W. T. Hanson, John McEncroe, W. G. Schermerhorn, Gerardus Smith.

*Official Headquarters and Place of Sessions.*—The Albany Historical and Art Society, 176 State St.

*Hotel Headquarters and Bureau of Information.*—The Ten Eyck, corner of State and Chapel Sts.

The opening session was held on Tuesday evening, February 17th.

An address of welcome was delivered by Hon. Charles H. Gaus, Mayor of Albany, and appropriately acknowledged by President Olcott.

The President announced, with suitable comment, the death of Hon. Abram S. Hewitt, a member of the Institute since the year of its foundation, and twice elected its President.

Dr. A. V. V. Raymond, President of Union University, being invited to speak concerning the death of Mr. Hewitt, said:

“In each generation it is given to a few so to live that, when they die, a common sorrow sets anew the seal of human brotherhood. In places where their feet have never trod, in homes never brightened by their presence, comes a sense of loss like the shadow of a cloud—a passing cloud; for soon all that they were, all for which they lived and labored, shines again with clearer light in the memories that men cherish. Among the few who in this generation have belonged

not to one community, but to all communities, not to one class, but to humanity, is Abram S. Hewitt. Without pretense, without self-seeking, he became a leader in the thought and activities of his age. And the ever-widening circle of his influence showed in every part the touch of a great spirit, of a life ennobled and exalted by those aims that never fail to gain the recognition of reverent and grateful affection. He could not have won and kept his large place in the modern world without rare ability in practical affairs. That ability was his by inheritance and training. Given five talents, with them he gained other five talents. He could think clearly and comprehensively, could plan broadly and act aggressively. He could bring things to pass that others only dreamed; but he is remembered to-day not so much for his deeds as for himself—for the honesty that was never challenged—for the purity that was never tarnished—for the sense of honor that was never lost; and more than all, also, for the charity that never failed. The tribute which we pay to his memory is a tribute to the merchant and financier, the publicist, the patron of the arts and sciences, the statesman, the philanthropist, the Christian citizen, Abram S. Hewitt."

The following cable message from the Secretary was read:

"SYRACUSE, SICILY, February 16, 1903.

"Best wishes for the Institute and the Albany meeting. Regret especially inability to express personally, on that occasion, my love for Mr. Hewitt and admiration of his character. May the inspiration of his life rest upon us forever!

"R. W. RAYMOND."

Dr. Albert R. Ledoux read an abstract of a biographical notice of Mr. Hewitt, prepared by Dr. Raymond.\*

An oral account of a Reconnaissance along the Rocky Mountains of British Columbia was given by Howard W. Dubois, Philadelphia, Pa., who exhibited in illustration a large number of lantern-views.†

The second session was held at the same place, Thursday afternoon, February 19th.

President Olcott announced the death of W. J. Taylor, a valued member of the Institute since its foundation and at one time a member of the Council.

The following papers were presented in oral abstract by their authors and accompanied by numerous lantern-views:

Electrical Power-Transmission for Mines, by F. O. Blackwell, Schenectady, N. Y.

Application of Electricity in the Anthracite Coal-Field of Pennsylvania, with Special Reference to the Wyoming Field, by H. H. Stoek and G. W. Harris, Scranton, Pa.

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\* Published in full elsewhere in this volume. † Not furnished for publication.

After some discussion of Mr. Stoek's paper by Mr. R. V. Norris and Mr. Warren, Electrical Engineer for the Delaware, Lackawanna and Western Railroad Co., the following paper was read by the author and discussed :

Electrical Apparatus for Coal-Mining, by W. B. Clarke, Schenectady, N. Y.

The President announced the appointment, as Scrutineers to examine the ballots received and report the names of the officers elected for the ensuing year, of Messrs. T. A. Rickard, A. L. Walker and A. H. Fay.

The third session was held at the same place on Friday morning, February 20th.

The annual report of the Council was presented, as follows :

#### ANNUAL REPORT OF THE COUNCIL.

In accordance with the Rules, the Council makes the following report to the Institute for the year ending December 31, 1902 :

The financial statement of the Secretary and the Treasurer shows receipts from all sources for the year ending December 31, 1902 (including the balance of \$4,749.30 on hand December 31, 1901), of \$49,118.69, and expenditures of \$37,544.02, leaving \$11,574.67 cash on hand. No account is taken in this statement of the increased value of the assets of the Institute in back volumes of the *Transactions*, office furniture, etc. In addition to the cash on hand, the Institute possesses invested funds of the par value of \$15,900 and market value of more than \$20,000, yielding about \$950 interest annually; and on December 31, there were no bills payable or outstanding obligations.

The detailed statement is as follows :

#### RECEIPTS.

Balance from statement December 31, 1902,	.	\$4,749 30
Annual dues,	. . . . . \$28,921 59	
Life-memberships,	. . . . . 4,190 00	
Binding of <i>Transactions</i> ,	. . . . . 4,016 74	
Sale of publications,	. . . . . 6,115 24	
Electrotypes,	. . . . . 31 25	
Interest on bonds and deposits,	. . . . . 1,088 57	
Miscellaneous,	. . . . . 6 00	
	<hr/>	\$44,869 39
		<hr/>
		\$49,118 69

## DISBURSEMENTS.

Printing volume xxxi. of <i>Transactions</i> , . . .	\$3,224 25	
“ pamphlet edition of papers, . . .	3,529 49	
“ new volume on “Ore-Deposits,” . . .	382 69	
“ “Evolution of Mine-Surveying Instru- ments,” . . . . .	155 33	
“ circulars and ballots, . . . . .	203 85	
Binding volume xxxi. and miscellaneous volumes, . . .	3,054 55	
“ new volume on “Ore-Deposits,” . . .	618 46	
“ “Evolution of Mine-Surveying Instru- ments,” . . . . .	103 50	
“ exchanges, . . . . .	99 95	
Engraving and electrotyping, . . . . .	1,103 28	
Secretary's department, including clerks, stenog- raphers, and expenses of editing and proof- reading, . . . . .	9,122 00	
Postage, including post-office box-rent, . . .	2,437 26	
Stationery, . . . . .	927 20	
Rent, . . . . .	2,500 00	
Express and freight charges, . . . . .	1,606 90	
Telephone, . . . . .	169 20	
Telegrams, cablegrams and car-fare, . . .	74 93	
Office equipment, . . . . .	446 49	
Assistant Treasurer's department, . . . . .	4,550 00	
Storage of <i>Transactions</i> , . . . . .	141 62	
Special stenographers and expenses of meetings, . . .	158 95	
Office supplies and repairs, . . . . .	790 86	
Refunding payments, . . . . .	9 42	
Insurance, . . . . .	25 90	
Collection charges, . . . . .	13 34	
Extra clerical help, . . . . .	105 12	
Library additions and librarian, . . . . .	1,137 20	
Printing index, . . . . .	530 73	
Binding “ . . . . .	301 55	
Translations, . . . . .	20 00	
		<hr/>
		\$37,544 02
Balance in cash account, . . . . .	\$11,574 67	

The changes which have taken place in the membership during the year are shown in the table on page xxviii.

In this tabular statement, the reported membership at the date of the last annual report has been omitted, for two reasons: first, because the practically obsolete class of foreign members elected prior to February, 1880 (since which date all foreign members have been included in the general list, with all the obligations and privileges of other members), is now omitted from the published formal record; secondly, because the reported totals, made up from year to year by simply adding the

*Membership of the American Institute of Mining Engineers,  
January 1, 1903.*

	H. M.	M.	A.	Totals.
Gains: By Election.....	2	338	15	355
“ “ Change of Status.....		8	1	9
“ “ Reinstatement.....		2		2
Losses: By Resignation.....		24	8	32
“ “ Dropping.....		39	3	42
“ “ Change of Status.....		1	8	9
“ “ Death.....	2	27	1	30
Total gains.....	2	348	16	366
Total losses.....	2	91	20	113
Present membership.....	10	3,095	157	3,262

gains and subtracting the losses, had become seriously incorrect. This result was due to the payment of arrears of dues by members who had been temporarily dropped for non-payment, and the clerical failure to report all such instances as technically “gains.” The result was, that upon a careful count of members and associates, the total was found to be larger than that reported. The totals given above have been verified by actual count of the list of members and associates on January 1, 1903.

The list of deaths reported during the year 1902 comprises the following names (the figures in parenthesis indicate the year in which the persons named were elected to membership):

*Honorary Members.*—Manuel Maria Contreras (1902), W. C. Roberts-Austen (1899).

*Members and Associates.*—George A. Bell\* (1888), L. C. Bierwirth (1875), W. P. Butler (1900), Harold M. Cole (1896), Arthur L. Collins (1895), Aaron French (1894), Howard Van F. Furman (1880), Jesse Hall (1888), F. N. Holbrook (1875), Thomas A. Ireland (1890), Isaac G. Johnson (1879), William W. Lindsay (1902), Thomas S. McNair (1871), Paul Mellors (1899), Edgar H. Messer (1900), J. C. Parkes (1875), Josiah Pierce, Jr. (1900), John R. Powell (1899), Charles Q. Rawlings (1891), P. Reddy (1892), Henry M. Stanley (1889), Edwin A. Stevens (1899), William F. Stevens (1896), William Swindell

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\* Death not reported until after January 1, 1903, and, therefore, not included in the above table. Moreover, this name is retained in the list of members prefixed to vol. xxxii., and there said to have been corrected up to the date named.

1879), Charles R. L. Tucker\* (1897), William Van Slooten\* (1884), Henry A. Vezin (1874), J. Wheelock (1882), John S. Whyte (1898), Henry Williams (1875), James B. Young† (1879).

A biographical notice of Arthur L. Collins has been prepared separately and will be published in the *Transactions*. Concerning the remainder of the names in the above list, such data as the Secretary has been able to obtain are given below in the alphabetical order of the names,—except as to the two honorary members, whose names are placed first. Further data, secured hereafter, may be introduced into this report before final publication, or included in the next annual report.

*Manuel Maria Contreras* was born in Mexico, February 21, 1833. His father was Don José M. Contreras, Commissary-General of Artillery of the Republic of Mexico.

He made his first studies in the Spanish-Mexican college, where he distinguished himself so much by close attention to study that his parents conceived the desire of securing for him admission to the national mining-college, then a difficult matter, because that institution was reserved specially for the sons of miners. It was finally arranged, however, in 1846, that he might enter as an outside student. During the first year he was very fond of athletic exercises, in which he distinguished himself; but he was not for that reason the less studious and attentive to work; and he obtained in that year some prizes, and the honor of speaking in public, reserved for students eminent in scholarship. The Hall of Sessions—where the meetings of this Institute were held during the Mexican meeting of 1901—was the scene of his first triumphs in such public performances.

In 1847 he left college, to follow his military father in the struggle of the Mexican-American war. In 1848, after the termination of this war, he resumed his studies; and in 1854 he entered the mining-school of practice, just then founded at

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\* See footnote on preceding page.

† Mr. Young's death occurred some years ago, but was not reported to the Secretary's office; and, although communications addressed to him were returned by the Post Office as undelivered, it was not proper to drop his name for that reason, because he was a life-member.



Fresnillo, Zacatecas. The separation from his parents, for whom he had great veneration, was the cause of great grief to him; and soon after beginning his studies his father died, since which time, it seems, he developed his trait of taciturnity. During this latter part of his studies he gave signs of exceptional aptitudes, as he undertook technical studies outside of the curriculum in the school, such as the yield of the patio amalgamation, economical conditions of the concentration of ores, and the drawing up of the mineral plans of the Departments of Plateros and Fresnillo.

On March 14, 1856, he obtained his diploma as Mining Engineer and Metallurgist, and entered upon his first professional labors, in the camp of Temascaltepec, State of Mexico. At the end of that year the heirs of the Count of Regla placed him at the head of their important mining-affairs, in Real del Monte. In 1859, he was Manager of the Providencia mine, in the camp of El Chico, State of Hidalgo, and in the same year he was elected Mining Agent for the city of Pachuca. In 1860, the house of Jaecker put him in charge of the Guadalupe mine in Pachuca, and of the Purisima Grande reduction-works there. In 1862 he was called to Guanajuato to take the technical management of the mining-properties of Sr. Pérez Gálvez, owner of the oldest and most valuable mines of Guanajuato; and for two years he conducted with signal success this important enterprise.

On May 26, 1866, he was appointed Metallurgist of the Real del Monte Co. In 1868, he became Assayer of the Mint of the City of Mexico. In 1872, the ruinous condition of the Real del Monte Co. caused him to be recalled and appointed Consulting Engineer of that very important company, a position which he held until the day of his death. That company may well deplore his death, as the services he rendered to it for almost thirty years have been universally recognized. His luminous reports and patient investigations of the company's mines resulted more than once in the discovery of important new *bonanzas*, among them the one called *La Dificultad*, in the Real del Monte district, which proved of paramount value to the company.

Sr. Contreras prepared the Project of the Mining Code, unifying the mining-legislation of Mexico, which was put into

effect November 22, 1884. In 1886, he co-operated in shaping the changes to be made in the Department of Public Instruction. In 1874, he was Professor of Physics in the Preparatory School of Mexico, and for many years thereafter Professor of Mathematics also. He prepared a series of mathematical books which have been in use as text-books ever since. The subjects covered by this series, with which all the profession here in Mexico is acquainted, are arithmetic, algebra, geometry, elementary mechanics and trigonometry. All of the high schools of the republic use his text-books to-day.

He was President of the Chamber of Deputies in a very difficult time for the country, and he died a Senator of the Republic, as well as President of the Academy of Science of the City of Mexico.

In 1879 and 1880, Sr. Contreras was Chairman of the Public Works Department of the City of Mexico. He instituted in the city very important works for the prevention of damage by floods, before the completion of the final system of drainage. He introduced fundamental improvements in the water-works-service of the city, and made profoundly useful studies upon the question of city-pavements. He was a member of the Commissions to study the drainage of the valley of Mexico and the sanitation of the city, and his influence was decisive in carrying to completion the drainage-works of the valley. Excessive labors during all his life caused his eyesight to become very weak during his latter years.

The principal characteristics of his nature were an extreme modesty, a constant and well-marked sentiment of honor and justice, and a notable command of clear and concise language, indicating a corresponding lucidity and thoroughness of thought. He was an almost paternal counselor of a whole generation of young men, whom he initiated into practical life, not only giving encouragement and guidance, from his own knowledge and experience, but also using his influence to secure for them profitable and honorable positions in connection with the great mining and metallurgical enterprises of Mexico. He was an enthusiastic believer in his country's progress, a man of faith and undying energy, of a most excellent heart, kind and tender to his children, faithful to his friends, and a slave to the fulfillment of his duties. He died March 29, 1902, of an attack of the grippe.

At the third session of the Mexican meeting of this Institute, held in the Hall of Councils of the School of Engineers of the City of Mexico (a place in which, more than half a century before, Sr. Contreras had won his earliest laurels as a mining-engineer), the President proposed, on behalf of the Council, the immediate election to membership of Sr. Don Manuel M. Contreras, "mathematician and engineer, formerly Mayor of the City of Mexico, now a Senator of the Republic, and long connected with the School of Mines." The unanimous election of the candidate was emphasized by a rising vote, and acknowledged by him in a felicitous speech.

As soon as practicable after the Mexican meeting, this distinguished member was duly proposed for the further recognition of honorary membership; and his name, unanimously endorsed by the Council, was submitted to the membership of the Institute for election by postal-ballot. The result is shown in the following letter:

"NEW YORK, May 21, 1902.

"SEÑOR DON MANUEL MARIA CONTRERAS,

"President of the Academy of Science,

"City of Mexico, Mexico.

"*Dear Sir:* I have the honor and pleasure to inform you that upon the cordial recommendation of the Council, and by unanimous vote (declared through postal-ballot) of the Members and Associates of the American Institute of Mining Engineers in the United States, Canada, Mexico and many other countries, you have been elected an Honorary Member thereof, in recognition of your distinguished services to the sciences and industries connected with mining, and of the high esteem in which you are universally held by your colleagues throughout the world, as well as by your own countrymen.

"A suitable Certificate of Honorary Membership will be forwarded to you shortly. Meanwhile, we send you a copy of our new special volume on 'Ore-Deposits,' and beg to say that the pamphlets and volumes of the Institute will be regularly forwarded to you hereafter, as they are issued.

"Yours, truly,

"R. W. RAYMOND,

"Secretary."

It will be seen that, before this letter was written—indeed, before the election took place—the distinguished patriot and engineer, by honoring whom the Institute had sought to honor itself, had been removed by death beyond the sphere of earthly ambitions, achievements and rewards. The certificate of his honorary membership was received by his family and cordially acknowledged by them as a testimony to be treasured for future years and generations.

*Sir William Roberts-Austen*,\* K.C.B., was born in 1843, entered the Royal School of Mines, London, in 1861. After his graduation, he became an assistant to the late Prof. Thomas Graham, Master of the Mint, with whom he was associated in many of the delicate and important researches with which that eminent chemist's name is associated. In 1869, upon the death of Graham, he became Assayer to the Mint, and in 1882 he was advanced to the position of Queen's Assay-Master. Meanwhile, he had been appointed in 1880, upon the retirement of Dr. Percy, Professor of Metallurgy at the Royal School of Mines,—a position which he held to the time of his death, in addition to his office in connection with the Mint, of which, in 1902, he became Deputy-Master. It is estimated that during his charge of the assay-department of that institution, he was responsible for the standard fineness of about £130,000,000 of gold coin, the accurate uniformity of which was universally recognized as a remarkable instance of scientific precision. On all questions connected with the technical operations of coinage, he was an acknowledged authority throughout the world.

His work as a lecturer and instructor at the Royal School of Mines led him, as the same work had led his illustrious predecessor, Dr. Percy, to make noteworthy contributions to the literature of metallurgy. His "Introduction to the Study of Metallurgy," of which the first edition was published in 1891, a second in 1893, a third in 1894, a fourth in 1897, and a fifth in 1902, was an admirable summary and guide in that department, and constituted the first of a series of volumes, two of which were written, and all were edited by him.†

But these labors, arduous and creditable as they were, do not constitute the chief basis of his fame, which rests rather upon the remarkable and important physical researches which he found time to make and to report, in the midst of duties onerous enough to excuse an ordinary man from such enter-

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\* His name was William Chandler Roberts, until 1887, when he added the name and arms of Austen, at the request of relations on the maternal side. He was knighted in 1889, in recognition of his eminent services to the British government and to science.

† "New Metallurgical Series," published by Charles Griffin & Co., and comprising, besides the work mentioned above, treatises on Gold (Rose), Iron (Turner), Steel (Harbord), Lead and Silver (Collins), Metallurgical Machinery (Jenkins) and Alloys (by Roberts-Austen himself).

prises. The following excellent summary of his achievements is taken from London *Engineering* for November 28, 1902:

"The work in this direction may be said to have commenced with Graham in the '60's, and was associated, amongst other matters, with the occlusion of gases in metals; but in 1880, when he succeeded Dr. Percy as Professor of Metallurgy at the Royal School of Mines, while still occupying his post at the Mint, his work broadened to the fullest extent; and he gave, in a paper in 1888, the results of experiments which determined the influence of impurities on the density and extensibility of gold, and established that this was due or controlled by the periodic law. An element, with a larger atomic volume than gold, greatly diminished both density and extensibility, whilst an element with smaller atoms tended to improve these qualities. This was probably the beginning of his long series of experimental research on the atomic theory of metals, and of the influences of traces of impurities on the whole mass. The Alloys Research Committee of the Institution of Mechanical Engineers was appointed in 1889, and there was thus provided a *modus vivendi*, and, it may be added, financial resources for the pursuit of such research-work. Roberts-Austen acted as "reporter" to the Committee, and practically carried out all the tests; and no one who heard him in successive meetings expounding the difficult problems associated with the chemistry and micro-structure of metals could fail to appreciate his disinterested enthusiasm and great energy and ability.

"This Committee has made five reports, all, with the exception, perhaps, of the first, of direct practical utility; and a final report has, we believe, been left in draft-form by Roberts-Austen. The first, submitted in 1891, was largely concerned with silver and gold and their impurities, with a continuance of the argument already referred to, that the properties of the elements are a periodic function of their atomic weight; but thus early he evolved the cooling-curve system in connection with alloys, which has since been developed so advantageously. In the second report, of 1893, we had a consideration of more practical metals. The effects of arsenic, antimony and bismuth on copper were discussed with a view to alloys of merchantable metal, while the thermal behavior of chromium steel was treated. The development of the next two years brought us some definite facts on the significance of the freezing-points of metals, with cooling-curves for electro-iron, and data on the effects of alloying aluminum with iron, copper and nickel, now utilized for motors for bicycles and cars, if not also in the machinery of very high-speed vessels of the torpedo type. Brasses, coppers and the diffusion of metals were the main points of the 1897 report, with a further consideration of the relation between the melting-points of alloys and the atomic volumes of their constituent metals, and as a development of this the freezing-point effect on copper-tin alloys. It was about this time, too,—in 1896—that he awakened popular interest by his establishment of the diffusion theory by popular demonstration, in his Bakerian Lecture at the Royal Society, when he showed that, with a block of gold and lead, laid closely together with a mathematically-accurate contact, the atoms of lead would pass into the gold. That the work of the Research Committee was adapted to the concurrent requirements of the metallurgist is proved by the fact that the fifth report dealt with carburized iron at a time when interest was directed to the influence of cementation and chilling for the hardening of armor-plates. In this 1899 report various alloys were discussed, and the treatment of low-carbon rail-steel was considered from many points of view. There remains the final report of the Committee, already prepared; it

deals with the annealing of steel in a characteristically comprehensive way, and an appendix on the microscopic alloys of copper and tin has already been presented separately to the Institution. The work of the Committee will now be continued in the National Physical Laboratory, for the establishment of which Roberts-Austen was a vigorous advocate; but there can be no question that the research undertaken, and the light it has thrown on the molecular changes in alloys, has already greatly extended their use.

"It goes without saying, that the extent and nature of the work demanded considerable resource; and a typical case may be quoted in the introduction, by the subject of our memoir, of an automatic recording-pyrometer. La Chatelier had invented an instrument for the purpose, but it lacked the important essential of recording autographically the temperatures; and Roberts-Austen brought photography to his aid for this purpose. This he described in papers at the Iron and Steel Institute, in 1891, 1892 and 1893. To the same Institution he read a paper on the rate of diffusion of carbon in iron, in 1896, bearing on the same subject as the alloys research-work; and, while working under the same auspices on carburization, he contributed to the Proceedings of the Iron and Steel Institute and to the Royal Society memoirs on the carburization of iron by a diamond. Indeed, with a facile pen, in the hands of a master of such an absorbing topic as the importance of alloys and the influence of molecular action, and equipped with facility for making tests and taking micro-photographs, it would have been surprising if records were not made in the Proceedings of almost all our institutions. Thus, alloys formed the topic of his British Association lecture at the Newcastle meeting in 1889, dealing with the hardening and tempering of steel, which was the means of beginning that long association with Osmond, the great French metallurgical chemist, that has added so much to our knowledge on the question of the thermal treatment of metals. It also formed the theme of his Cantor Lectures to the Society of Arts; of frequent addresses at the Royal Institution, where his skill as a demonstrator made his performances as welcome as they were profitable to the audiences; of a paper to the Institution of Civil Engineers, and of this year's James Forrest Lecture to the same Institution. On each occasion a new phase of the subject was presented, and that with great perspicuity.

"For the Iron and Steel Institute he did splendid service; and, in addition to the papers to which we have specifically referred, he made a notable triumph in his address on the action of projectiles and of explosives on gun-tubes, delivered at the Stockholm meeting, when the King of Sweden was present. The micro-photographs of gun-tubes were strikingly beautiful. He was President in the year 1899-1900, and his address at the spring meeting of the former year was a masterly review of the work of his predecessors, while at the Paris meeting, in 1900, he similarly eulogized the work of the French scientists. Both addresses were scholarly in style, and of a most informing nature. Indeed, they have been referred to as exceptional in these respects. It was during his presidency, and perhaps more than a coincidence, that Dr. Andrew Carnegie offered scholarships of research to the Institute. Roberts-Austen, too, improved the status of the Institute by enlisting the sympathy of Royalty. It was when he was President that a Royal Charter was granted, and it was from his hands that the Bessemer medal was graciously accepted by her late Majesty, Queen Victoria.

"Roberts-Austen rendered service on several Government committees. In 1893, he was chosen to act as Chairman of a committee appointed to inquire into the laboratory arrangements of the Customs and Inland Revenue Departments. In the same year he served on a committee, appointed by the Secretary of State

for India, to consider the best means of utilizing for metallurgical purposes the water-power available on the completion of the Periyar Water-Works. In 1896, he was a member of the Board of Trade Committee appointed to consider the cause of the deterioration of steel-rails in ordinary use. In connection with the committee, he conducted an elaborate research and furnished a report of great industrial importance. In 1897, he was directed to serve on a committee appointed to consider the desirability of establishing a national physical laboratory, and was chosen as a member of a small committee to report on the most suitable steel for ordnance work.

"Many honors, the highest his profession could pay, fell to his lot. He was President of the Iron and Steel Institute in 1899-1901, was elected an honorary member of the Institution of Mechanical Engineers in 1897, and of the Institution of Civil Engineers in 1901. He was one of the founders of the Physical Society of London, of which he was for some time secretary and afterwards a vice-president, and was an honorary secretary of the British Association for the Advancement of Science. He was elected a Fellow of the Royal Society as early as 1875, and served on the Council. He was also a vice-president of the Chemical Society and of the Society of Arts. He was made a C.B. in 1890, and a K.C.B. in 1899, a D.C.L. of Durham College in 1897, and a D.Sc. of Victoria University, Manchester. He also served on the Government Commission in connection with various exhibitions inventions, Paris, 1889, and Chicago. He was a Knight of the Legion of Honor, and was elected, in 1893, a member of the Athenæum Club, for distinguished eminence in science."

Roberts-Austen's views and experiments have been repeatedly cited in the *Transactions* of the Institute; but his only direct contributions were a memorable paper on "Recent Advances in Pyrometry," presented at the Chicago meeting of August, 1903 (*Trans.*, xxiii., 407), and an intelligent, friendly criticism (*Trans.*, xxvii., 860) of Mr. Sauveur's paper on "The Micro-structure of Steel and Theories of Hardening" (*Trans.*, xxvi., 863). In 1899, he was elected an Honorary Member, in recognition of his eminent services to the science of metallurgy.

His death, at the age of 59, in the prime of his matured experience, skill and reputation, cut short a career of distinguished honor and usefulness, and deprived the professional and scientific world of unknown treasure which his further activity might have been expected to bestow. But the unwearied industry with which he recorded and reported each step of his progress has made the results of his work our property, to be gratefully marked henceforward with the name of its unselfish donor. We have not to lament, as we are so often called to do, the loss of knowledge already possessed by one who might have made it ours, as well as his. What he had he gave promptly, as well as freely, and death was not able to deprive us of all the harvest of his fruitful life.

He was buried at Canterbury, November 27, 1902. A memorial service was numerously attended by personal friends and representatives of the societies and institutions with which he was connected. And a service, not less significant, took place at the beautiful mission church which he had erected near his house at Chilworth, Surrey, in which, as diocesan reader, he was wont to minister.

*George A. Bell*, who died August 26, 1902, was born in Brooklyn, N. Y., September 6, 1851, and educated in the public schools and in the Polytechnic Institute of that city. He began his work as a boy of 16 in a commission-house in New York, but soon went into the employ of the Pratt Oil Refinery, shortly afterwards merged in the Standard Oil Company. Here he rapidly advanced, and, after having charge of the office of the Company for years, went, in 1887, to Troy, N. Y., as auditor of the Troy Steel and Iron Company.

Within a year he was made general superintendent, his duties including entire management of the works, contracting for material and supplies and supervision of the manufacturing accounts, during the period in which the large product of the plant was changed from rails to billets. When the Company underwent reorganization, Mr. Bell was prominent in all negotiations, and was sent abroad to inspect methods of steel-manufacturing in England and Europe. In 1900, Mr. Bell resigned his position and removed to Marion, Ind., where he had bought an interest in the Malleable Iron Works, and held the position of treasurer of the Company. His career was cut short by his sudden death in the office of the works.

Mr. Bell was a kindly and genial man, and both in his business relations and in divers social organizations held the respect and friendship of many. His membership in the Institute dates from 1888, when he became superintendent of the Troy works.

*Leopold C. Bierwirth*, who was for years the secretary and general agent of the New Jersey Iron Mining Co., of Dover, N. J., died at his home in that city on February 3, 1902. Mr. Bierwirth was born in New York City, June 5, 1838, and at the age of 16 he entered the Mining Academy of Clausthal, Ger-



many. Here, and at the University of Heidelberg, he passed three years studying mine-engineering and metallurgy, and complemented this with two years in England among the iron-furnaces and rolling-mills. He returned to America at the age of 21, and entered the employ of the Trenton (N. J.) Iron Company. Five years later, in 1864, he was employed by parties interested to purchase the mining-lands which formed the basis of the New Jersey Iron Mining Co. He was made secretary and general agent of the company from the outset, and retained the position until the time of his death. Mr. Bierwirth joined the Institute in 1875.

*Aaron French*, President of the A. French Spring Company, of Pittsburg, Pa., and also prominently connected with the recently-organized Spiral Spring Company of America, died at his residence in the East End, Pittsburg, March 23, 1902. His death came after about a month's illness, the result of a stroke of paralysis, at Atlanta, Ga.

Aaron French was one of the oldest men actively engaged in the manufacturing business in Pittsburg. He was born at Wadsworth, Medina county, Ohio, March 23, 1823. His school education ceased when he was 12 years old, at which time he began work on a farm. When he was 13, he learned blacksmithing, a few years later going to Cleveland to work for the Ohio Stage Co. Later, he left blacksmithing, and was employed as Western agent by the American Fur Company. About 40 years ago he went to Pittsburg, and it was then that the A. French Spring Co. came into existence. It was Aaron French who made the first of the steel springs now used by every railroad, and at the time of his death he was at the head of the largest steel-spring manufacturing industry of its kind in the world. He was also connected with other enterprises, among them being several gold-mining companies in the South. He joined the Institute in 1894.

*Howard Van Fleet Furman* was born in New York City, May 17, 1861. His death from pneumonia in Mapimi, Durango, Mexico, on April 28, 1902, was widely mourned, both in social and professional circles. Graduated from the Columbia School of Mines, in 1881, he went to Colorado, in 1882, as assayer for

the Globe Smelting and Refining Co.; and during the next six years was also, successively, chemist and foreman of the Germania Lead Works in Utah, assistant superintendent of the Billing's Smelter, Socorro, N. M., and superintendent of the Bailey Smelter, in Denver. From 1888 to 1894 he was occupied in expert-consulting work, principally in Arizona, and during the next year was chief-assayer of the United States Mint at Denver. Dr. Furman's studious industry kept him abreast with every development affecting his profession, and in the autumn of 1895 he took the Chair of Metallurgy in the Colorado School of Mines, at Golden, where for nearly two years he labored most successfully, with an influence on the students at once personally agreeable and professionally stimulating.

About this time, the Pentagon Co. of London, Eng., secured his services for a trip to Abyssinia, with the purpose of obtaining certain mining-concessions. Under the conditions imposed, however, this proved impracticable. An expedition to Alaska followed, on which he made an interesting and able report. In 1901, Dr. Furman accepted the position of general superintendent of mines for the Compañía Minera de Peñoles (mining, smelting, etc.), in Mapimi, Mexico. In February, 1902, he entered upon his duties there with his customary efficiency and thoroughness, combined with his personal attractiveness, which inspired both fondness and respect. His sudden death in the April following was felt as a severe loss.

Dr. Furman's professional activities did not prevent his preparation of many papers for scientific societies, and a "Manual of Practical Assaying," made by him, was an immediate and continuing success. He joined the Institute in 1880. A list of his contributions to the *Transactions* is appended.

Title.	Transactions.	
	VOLUME	PAGE
<i>Losses of Gold and Silver in the Fire-Assay,</i> .	xxiv.	735
Remarks on a uniform method for assaying copper-materials, . . . . .	xxiv.	874
<i>The Assay of Silver Sulphides,</i> . . . .	xxv.	245, 999
Remarks on the inaccuracy of the commer- cial-silver assay, . . . . .	xxv.	871

<i>Title.</i>	<i>Transactions.</i>	
	VOLUME	PAGE
Remarks on assay of copper and copper-matte, . . . . .	xxv.	1002
<i>Laboratory-Tests in Connection with the Extraction of Gold from Ores by the Cyanide Process,</i>	xxvi.	721
Remarks on the occurrence and behavior of tellurium in gold-ores, . . . . .	xxvi.	1106
Remarks on discovery of new gold-districts, . . . . .	xxix.	1038
Remarks on plate-amalgamation, . . . . .	xxix.	1042
Remarks on pyritic smelting in the Black Hills, . . . . .	xxx.	1125

*Jesse Hall* was born in Camden, N. J., in the year 1849, and there reared. At 20 years of age he went to Phoenixville, Pa., as a mechanical engineer, and until the time of his death held a responsible position in the engineering department of the Phoenix Iron Company, where he enjoyed full confidence and esteem. He died February 16, 1902. He became a member of the Institute in 1888.

*Francis Newberry Holbrook*, born in Cleveland, O., June 22, 1854; died in New York City, August 12, 1902. In 1862, he went to Brooklyn, N. Y., where, in the Juvenile High School and the Polytechnic Institute, he was prepared for the School of Mines, Columbia University, from which he was graduated in 1876 with degree of M.E. For several years he remained at the School of Mines as assistant in the laboratory, leaving there in 1880 for a trip to Europe, followed by a governmental geological surveying-expedition in western Texas. After this he was, until 1884, superintendent of the Corralito Co., Chihuahua, Mexico; from 1884 to 1887, he conducted business in El Paso, Texas, as consulting expert; in 1887, he became superintendent of United Verde Copper Co., Arizona; in 1888, he was employed in geological work for the Southern Pacific R.R. Co. in western Texas; after which he took charge of a gold-mine and mill in Arizona on his own account. In 1890, Mr. Holbrook entered the service of the Compañia Metalurgica Mexicana, in which, with brief intermissions, he continued until his illness and death in 1902. Besides the Institute, which he joined in 1875, he was a member of the Society of Mechanical

Engineers, and of the Reform Club and Mendelssohn Club of New York City.

*Thomas A. Ireland* was widely known in Colorado as a ranch-owner and mine-operator, his residence being divided between Denver and New York City. He was born August 1, 1854, and died after a brief illness, September 18, 1902, in Cleveland, O., the home of his wife's family. At the time of his last illness he had been preparing, after some years of comparative leisure, to re-engage actively in mining-operations on his properties in Clear Creek county, Colo., with a newly-organized company. He had been a member of the Institute since 1890.

*Thomas S. McNair*, born 1824, in Hanover, Pa., studied at Williams College, Mass., and subsequently taught school for a number of years, but ultimately adopted the profession of engineering. He was connected with the construction of the Raritan and Passaic canal from Easton to Perth Amboy; was assistant division-engineer under W. F. Shunk (one of the most famous men of his time in that line) in the building of the North Penn railroad between Bethlehem and Philadelphia; and was also employed as an engineer in the construction of the Lehigh canal.

About 1854, while still a young man, he made Beaver Meadow, Pa., his residence. Somewhat later, together with the late W. R. Maffet, he built that part of the North Branch canal which, extending from Wilkes-Barre to the New York boundary, was subsequently transferred to the Lehigh Valley Railroad Co., and abandoned. Many such early internal waterways, with which Mr. McNair and other engineers had so much to do, have long since become of subordinate importance, or have disappeared altogether, in competition with later means of transportation. But it should not be forgotten that, in their time, they aided mightily in the commercial development of the country, and that their constructors were the leaders of American engineering.

A little later, Mr. McNair moved to Hazleton, Pa., where he resided until his death. The occasion of this change of residence was his acceptance of the position of chief engineer of

the Hazleton, subsequently merged into the Lehigh Valley, railroad. Under both administrations, he retained his position until 1893. Under his direction the tunnel between Hazleton and Jeddo and the link between Hazleton and Pottsville were constructed. The Harleigh canal and the famous Jeddo tunnel are witnesses of his skill as an engineer.

*Paul Mellors* was born at Locksley House, Nottingham, England, August 25, 1867. After entering upon active work, he was seven years with a firm in Nottingham, and then, coming to the United States, was three years in real estate business in Denver, Colo. He was one of the enterprising but premature originators of the Colorado sugar-beet enterprise. After this, Mr. Mellors ranged wide in his labors, being engaged in business in Chicago and London; manager of Golden Cross mines in Western Australia; reporting on property in Tombstone, Arizona; managing copper-mines in Southern Spain; reporting on mining properties in Northern Spain, Siberia, Bavaria and Cape Coast Colony in West Africa. In this latter region he spent seven months, returning to England in June, 1902. He was taken ill in October with influenza, typhoid fever and cerebro-spinal meningitis, and died December 11, 1902. His Institute membership dates from 1899.

*Edgar H. Messer*, who died August 12, 1902, had but shortly before that joined the British Engineers' Alliance, Limited, of Johannesburg, Transvaal, as chief engineer. Much was expected of him, from his notable ability and his thorough knowledge of every detail connected with the mining industry. With the exception of a nine months' visit to the Australian gold-fields in 1896-97, Mr. Messer had for many years been assistant mechanical engineer of the Consolidated Gold-Fields of South Africa, Limited, under Mr. J. B. Pitchford and, later, Mr. H. C. Behr, chief engineers, both of whom bear high testimony to his services and worth. He joined the Institute only in 1900.

*J. C. Parkes* was born in England in 1831, coming to America with his parents the following year. His education was furnished by the common schools, and as a boy he worked with his father in the rolling-mills. In 1850, he was thus

placed in the mills at Wheeling, Va., whence he went to California, remaining there nine years, prospecting and mining, and, in 1859, returning to the Eastern States. The following year, however, he went back to California, and for five years conducted a successful newspaper in one of the mining-towns. Forced by ill-health to sell out and leave the Pacific coast, he secured employment in Chicago with the North Chicago Rolling-Mill Co., and after a few years—in the early '70's—was made superintendent of the mills on the North branch; in '78, superintendent at Bay View, Wis.; and in '81, general manager of the entire system, a position which he held until the formation of the Illinois Steel Co. He served this company one year as superintendent of the North and South Chicago plants, and in 1889 withdrew, to enter the iron and steel commission business. He retired from active business about 1895, and died December 24, 1901 (knowledge of his demise not reaching the Secretary's office until too late for entry in last year's record). Mr. Parkes joined the Institute in 1875.

*Josiah Pierce, Jr.*, a Major of Engineers on the staffs of Generals Grant and Henry in the Spanish War, died in Washington from typhoid fever, in August, 1902. Major Pierce was born in Russia in 1861, his father being connected at that time with the United States Legation at St. Petersburg. He obtained his early education in Russian schools, and subsequently was an associate of King's College, London. He studied at Cambridge University and received the degree of master of arts from that institution. Upon coming to this country he took a post-graduate course at the Massachusetts Institute of Technology and also at Johns Hopkins. He served as instructor and professor of engineering in the Columbian University and in the Catholic University of America. He also performed some important work for the Geological Survey and the Coast and Geodetic Survey. In 1888, upon the request of the American Minister to London, he was permitted to make a special study of the method of ordnance survey in Great Britain, and won the Telford gold medal and the premium of the London Institute of Civil Engineers for a paper on "American Methods of Mapping." His membership in the Institute dates from 1900.

*John R. Powell* met his death, February 28, 1902, in efforts to save the lives of others, at the Liberty Bell mine, Telluride, Colo. He had studied two years at the Colorado School of Mines, and was intending to return the next year to complete his course, meantime holding a responsible position in the Liberty Bell mine, where both his employers and the men under his charge regarded him with confidence and affection. After a snow-slide had wrecked the ore-bins, upper tram-terminal and boarding- and bunk-houses of the mine, young Powell accompanied the first party of rescuers to arrive from the town, and was the last to leave, after doing everything possible to save imperiled lives; but on the way back to town he was himself overwhelmed in another snow-slide and perished. The year 1899 was the date of his joining the Institute, and, like many another hero in our profession, he fulfilled a man's duty in brief years.

*Charles Quincy Rawlings* was a very successful expert chemist, long known and trusted in the great manufacturing center of Wheeling, W. Va. Born and reared in that city, he was educated at the Linsly Institute, and for several years after graduation was retained there as instructor in chemistry. In 1887 he entered the Massachusetts Institute of Technology, and after completing his course there returned to Wheeling and began his professional career. He died March 5, 1902, at the age of 32. His membership in the Institute dated from 1891.

*Henry M. Stanley*, mining engineer and assayer, died of peritonitis, in the City of Mexico, December 18, 1902, at the age of 48 years. Mr. Stanley was one of the well-known and popular men of the American colony, having resided in Mexico for the past eight years, where he won for himself an enviable reputation, both as a mining engineer and assayer of note, and also as a thorough gentleman and social companion. He was one of the best-liked members of the American club. His home was in Cleveland, Ohio, where his parents still reside.

During the greater part of his career as a mining engineer, Mr. Stanley worked in the western United States. In 1896, he was in charge of the amalgamation in the Ojo de Agua mill, at San Luis de la Paz. In the following year he joined Mr. C.

W. Pritchett and opened an engineering and assaying office in Mexico City, which he took over in 1900, when Mr. Pritchett returned to the United States. He was an excellent chemist and assayer, and stood high, both professionally and socially. Financially, he was prospered, and was interested in a number of good mines. He joined the Institute in 1889, and contributed to its *Transactions* a very valuable and interesting paper on "Photographic and Co-ordinate Surveying,"\* which was presented at the Glen Summit Meeting, October, 1891.

*William Swindell* was born in Allegheny City, Pa., February 14, 1834, and died suddenly in Pittsburg, June 20, 1902. Mr. Swindell was the pioneer in furnace-construction in Pittsburg, and, upon the introduction of the regenerative gas-furnaces and gas-producers in this country, became the Pittsburg representative of Siemens Bros., under their general agents, Messrs. Richmond & Potts, of Philadelphia. He took great interest in the construction of these furnaces and contributed largely to their success. Moreover, he invented a number of improvements on the crucible melting-furnace which were adopted throughout the United States; and of the present furnaces of this type in operation among us to-day he constructed 90 per cent. At one time he had charge of the furnace-construction at every mill in Pittsburg.

He received for his inventions 66 U. S. patents in addition to a considerable number granted in Canada and England. Among his later inventions was a recuperative annealing-furnace, which is now in use in every important sheet- and tin-plate plant in the country; 300 of such furnaces having been constructed by him in less than 4 years. When the works of the American Tin-Plate Co. at Elwood, Ind., the first important tin-plate plant in the United States, was built, Mr. Swindell was called upon to construct the furnaces from original Welsh drawings, which he found to be crude and unsatisfactory. The annealing-furnaces had a capacity of only 18 tons. It was then that he invented his recuperative furnace, which has a maximum capacity of 90 tons per heat, and uses less fuel in annealing than the old Welsh 18-ton furnace.

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\* *Trans.*, xx., 740.



Mr. Swindell also constructed some 70 open-hearth furnaces; and his regenerative heating-furnace is largely used in rolling-mills.

He invented, moreover, a number of improvements in gas-producers, among them a water-seal, of which he built 1,100 in 4 years. Another, having a hot-air attachment, for use on rotary cement- and lime-kilns, is now being introduced by his successors, and is expected by them to revolutionize the methods of using fuel in these two industries.

Mr. William Metcalf, of Pittsburg, an ex-President of the Institute, and recognized as one of the highest authorities on steel-practice, writes to the Secretary concerning Mr. Swindell:

"He built all the large reverberatory furnaces used by the Fort Pitt foundry for the manufacture of heavy guns during the War of the Rebellion. For the Crescent steel-works, he built, without the aid of English expert representatives of Siemens, the first Siemens heating- and melting-furnaces erected in this region. He invented the "squirrel-tail" continuous regenerative gas-furnace, which was the best ever devised for puddling, and the best (except, of course, the Siemens) for heating.

"That he did not realize a large fortune from his inventions was due to the fact that he was a better mechanic than money-maker."

Like a host of other "self-made" engineers, Mr. Swindell realized the value of scientific knowledge as a preparation and aid to practical experience; and, in 1879, he joined this Institute, then comparatively young and numerically feeble, in a spirit of loyal co-operation with its aim and purpose, to which his uninterrupted membership of 23 years bears confirmatory witness. The debt owed by the Institute to the hearty co-operation of such men cannot be too often or too emphatically acknowledged.

Mr. Swindell "fell in harness," being actively engaged in his business up to an hour and a half before he died, leaving behind him the record of an upright, useful, fruitful life, and a character which received, as it deserved, the affection and respect of all.

*Charles Rupert Langdale Tucker*, concerning whose early life and career information is not yet available, was, at the time of his fatal illness, filling the position of assayer for the Mexican Ore Co., Jimenez, Chihuahua, Mex. Suffering from some apparently slight trouble, which, however, demanded an opera-

tion, he went to St. Luke's Hospital, El Paso, Texas; but typhoid fever gradually developed, and from its effects he died, August 2, 1902. His Institute membership dates from 1897.

*William Van Slooten* was the son of John Van Slooten, a German patriot, who, after the defeat of the Revolutionary movement in 1848, came to this country as one of the political refugees who furnished so valuable an element to American political life. Born in New Orleans in 1858, William Van Slooten studied at the University of Virginia, where he secured, at the age of 19, his degree as a mining engineer, thus receiving the distinction of being the youngest graduate in the history of the institution. He then studied law, but soon decided to engage permanently in his profession of mining engineering. Shortly before reaching his majority, he was appointed assistant-assayer of the U. S. Mint in New Orleans, and held the place for two years. For several years thereafter, he was engaged in mining-operations in New Mexico, Colorado, Arizona and Nova Scotia, representing in the last-named region a London syndicate. In 1881, he established himself as consulting engineer in New York, where he was at one time connected with the firm of U. O. Crane & Co., metal-dealers. At the time of his death he was President of the South American Development Co., in the service of which he had recently visited South America, where he contracted a fever resulting in his death, December 10, 1901.\*

*Henry Augustus Vezin*, who died suddenly from *angina pectoris* at Denver, Colo., December 27, 1902, was born at Philadelphia in 1835. At the age of 15, he went to Germany to complete his education, and, after his graduation abroad, he traveled extensively, studying the machinery and methods of mining, metallurgy and ore-dressing. Returning to the United States, he began his career in Philadelphia, first as a mining engineer and later in metallurgical work. But at the outbreak of the War of the Rebellion, in 1861, he joined the Fifth Pennsylvania Cavalry, and served throughout the war, attain-

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\* SECRETARY'S NOTE.—Since the above report was completed, the widow of Mr. Van Slooten has presented to the Library of the Institute, in his name, a valuable collection of more than 600 professional books, which had belonged to him.—R. W. E.

ing the rank of captain, together with brevets as major and lieutenant-colonel, "for gallant and meritorious service."

Returning to civil life at the close of the war, he settled first at Central City, Colo., where he built works for the treatment of the ores of Gilpin county by the Monier process of roasting and lixiviation. In 1872, he was connected with the St. John smelter, in Summit county, Colo., where he turned out what is believed to have been the first silver-lead bullion produced on a commercial scale in Colorado. Still later, he was employed in professional work (chiefly as designer of plants and machinery) at Leadville, Pueblo, etc., in Colorado, and also at various places in Montana, Utah and other Rocky Mountain States and Territories.

His most important foreign work was performed for a French company on the first Nicaraguan canal-survey. He also made for Philadelphia capitalists an extensive examination of Russian deposits of coal and of iron-ore.

After 1888, he made his headquarters at Denver, Colo., where he carried on a large business as a consulting engineer, his specialty being the preparation of plans and designs for smelters, sampling-mills and concentration-works. He invented a mechanical sampler which has proved successful in practice; and his thorough knowledge of mechanical engineering is embodied in many constructions which do not bear his name.

Partly through unnecessary modesty, partly by reason of engrossing occupation, and partly because he preferred to study rather than to teach, and to put his remarkable knowledge of technical literature at the service of others rather than to use it for his own credit, Mr. Vezin published few articles or treatises. He was a member of the Colorado Scientific Society and the North of England Institute of Mining Engineers, and of this Institute, which he joined in 1874, a time when the advantages of membership in it were far less clearly understood than they now are, and when the unusual range of its membership (including common miners, foremen, etc.,—in fact, all who were practically connected with mining and metallurgy) was somewhat distasteful to many educated engineers, who did not care to be associated with their professional inferiors.

During more than twenty-eight years of membership he remained loyally attached to the Institute, supporting on all

occasions its purpose and management. His contributions to the *Transactions*\* are simply discussions of the papers presented by other members; but they indicate so thorough a technical and historical knowledge of the subjects concerned (the first being the Colorado gold stamp-mills, and the second the construction of modern silver-lead smelting-plants) as to arouse keen regret that the accomplished author did not bequeath to his professional brethren a more complete summary of his wide professional experience.†

But, while we lament our loss, we must rejoice at our gain. Henry A. Vezin's whole life was one of generous co-operation with his brethren. Nothing that he knew was held as a secret for personal gain. He was ready to incur both labor and expense to solve a professional problem for a friend; and his quiet, unostentatious way of ignoring all sordid considerations, when such help was called for, is an example which few, perhaps, habitually imitate, but all must cordially admire.

*Jerome Wheelock*, whose name is notably associated with the engine of his invention, dropped dead in the street near his home in Worcester, Mass., on February 26, 1902. He was 67 years of age, and apparently in full health, his massive head, strong, genial features, keen eyes, and white hair and beard crowning an impressive figure, and making him a noticeable personality in the community. Born in Grafton, Mass., a poor boy, he ran away from home and sought a mechanical training as apprentice with the Taunton Locomotive-Works, whence he went to the Washburn Iron-Works at Worcester. While in the latter employ he invented the Wheelock cylinder steam-packing, which he began manufacturing in 1865, and which is still in use. In 1870, with the capital derived from this and other minor inventions, he opened a shop, where for twenty

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\* *Trans.*, xxiii., 547, and xxvi., 1095.

† SECRETARY'S NOTE.—This, I have reason to believe, he would have done, had he not been surprised by death. Now, all that he knew (and was so ready to place privately at the disposal of others) has perished with his departure, except a few hints, which show us what we have lost. I fervently wish that this lesson might not be lost upon the numerous members of the Institute who are waiting for the day when they can write something comprehensive and monumental, instead of being willing to assist the rest of us with contributions from their experience as they go along!—R. W. R.

years he labored, and where he invented and perfected the valve-mechanism which is the essential feature of the Wheelock steam-engine. This machine bore away the honors at the American Institute in 1875, at the Centennial Exposition of Philadelphia in 1876, at the Paris Exposition of 1878, and also won high encomium in London, Edinburgh and Brussels. A few years ago Mr. Wheelock left the manufacture of his engines to a company, and devoted himself to developing his patents in other matters. He was a man whose career conferred honor upon the city of his residence, and benefits upon thousands who never knew aught of him but his honored name. He became a member of the Institute in 1882.

*John S. Whyte* was born in Scotland in 1852. His parents removed to Canada during his boyhood, but he returned to Glasgow in 1872 to finish his studies. For several years after coming back to America, Mr. Whyte was in the employ of the Orford Copper Co. of New York, and traveled extensively for several years, visiting nearly every copper-mine of importance in the West. Following this, he represented the Nichols Chemical Co. of New York, first in Arizona and afterwards in Tamaulipa, Mexico, where its large copper-mines were situated. Severing this connection in 1899, Mr. Whyte visited all the mining-districts of northern Mexico, and was employed in negotiating for the various mining interests of the Torreon smelter. He died at Torreon, Mex., August 18, 1902, the last position he held being that of superintendent of the Jimulco Mining Co. He was well known and highly trusted throughout northern Mexico, and while there made some profitable investments in mining property. It was in 1893 that Mr. Whyte joined the Institute.

*Henry Williams*, a pioneer citizen of Butte, Colo., who died in that city July 8, 1902, was born in Truro, Cornwall, August 30, 1840. His early life was spent in mining and smelting near his home, and at the proper age he entered the Truro School of Mines, and later the Royal School of Mines in London. Afterwards he was a student at the Mining and Metallurgical School at Clausthal, Germany. Returning to England, he was employed for several years by Vivian & Sons, at Swan-

sea, for whom he acted later as ore-buyer at Mazatlan, on the west coast of Mexico.

Still later, he formed in England a smelting-company to operate in Clear Creek county, Colo. But, meeting strong competition from the Boston & Colorado Smelting Co., then located at Black Hawk, in an adjoining county, the works were given up in 1875, and Mr. Williams soon after accepted the management of the branch-works of the Boston & Colorado Co., at Alma, Colo. In 1878, smelting operations were given up at Alma, as the building of the Denver and South Park railroad made it practicable to ship ore to the main works of the company at Black Hawk, and the branch was continued only as a sampling-works.

Early in 1878, the Black Hawk works began to receive large shipments of copper-silver ores from the Gagnon and the Original mines in Butte, and Mr. Williams was employed by the Boston & Colorado Co. to visit the Butte district and report upon its prospective resources. The result was, that during the winter of 1878 and 1879 a company was formed called the Colorado & Montana Smelting Co., of which Mr. Williams became manager; and in the spring of 1879 he returned to Butte, purchased the present site of the Colorado smelter, and began the erection of suitable furnaces.

The importance of this enterprise, as a factor in the development of Butte's resources, can hardly be overestimated. The proceeds of ore sold to this smelter enabled claim-owners to continue development; and the formation of the Parrot, Montana Copper and Anaconda companies followed in rapid succession. These companies soon prepared to reduce their own ores, and thus the present enormous metallurgical industry of Butte was founded. Mr. Williams's own company, reorganized as the Boston & Colorado Smelting Co., acquired the Gagnon and other mines, and successfully maintained its position as a technical enterprise among its formidable rivals.

Close application to business in the early years of the company had its effect upon Mr. Williams's health, and his failing strength led him to resign two years ago. But he continued to live in the home which he had built at the smelter, and gave his time to many outside affairs in which he was interested, among them a large ranch in Deer Lodge county, placer- and

quartz-mines near Bearmouth and Philipsburg, in Granite county, a wood-flume in Silver Bow canyon, and other interests.

Mr. Williams joined the Institute in 1875, but was too deeply absorbed in his daily labors and responsibilities to contribute to its *Transactions*. The hospitable welcome and professional aid which he freely extended to its members, however, bore witness to the high esteem in which he held the Institute.

The Scrutineers, appointed for this purpose under Rule V., reported that the following officers had been elected:

*PRESIDENT.*

DR. ALBERT R. LEDOÛX, . . . . . New York City.

*VICE-PRESIDENTS.*

(To serve two years.)

JOHN MARKLE, . . . . . Jeddo, Pa.  
 PHILIP W. MOEN, . . . . . Worcester, Mass.  
 JAMES F. KEMP, . . . . . New York City.

*MANAGERS.*

(To serve three years.)

HEINRICH RIES, . . . . . Ithaca, N. Y.  
 B. B. LAWRENCE, . . . . . New York City.  
 F. KLEPETKO, . . . . . New York City.

*TREASURER.*

THEODORE D. RAND, . . . . . Philadelphia, Pa.

*SECRETARY.*

ROSSITER W. RAYMOND, . . . . . New York City.

Mr. W. L. Saunders, New York City, discussed, at length, Mr. Clarke's paper on Electrical Apparatus for Coal-Mining.

The following papers were presented in oral abstract by the authors:

The Cost of Pumping at the Short Mountain Colliery of the Lykens Valley Coal Co., by R. V. Norris, Wilkes-Barre, Pa.

Water-Hoisting in the Pennsylvania Anthracite Region, by R. V. Norris. Wilkes-Barre. Pa.

The following papers were read by the authors :

Electrolytic Lead-Refining, by Anson G. Betts, Troy, N. Y.

Notes on Accidents Due to Combustion within Air-Compressors, by Dr. Albert R. Ledoux, New York City.

Mr. Theodore Dwight, Acting Secretary, read the following papers by title :

The Determination of Power for Rolling Iron and Steel, by Louis Katona, Resicza, Hungary.

Flue-Dirt and Top-Pressure in Iron Blast-Furnaces: A Study of the Influences Controlling Them, by F. Louis Grammer, Baltimore, Md.

A Test for Precious Metals in Cyanide Solutions, by Albert Arents, Alameda, Cal.

Notes on the Metallurgy of Copper of Montana, by H. O. Hofman, Boston, Mass.

Note on the Influence of the Rate of Cooling on the Structure of Steel, by Albert Sauveur and H. C. Boynton, Cambridge, Mass.

The Effect of Fineness of Grain on the Fusibility of Clay, by Heinrich Ries, Ithaca, N. Y.

Notes on the New Jersey Fire-brick Industry, Heinrich Ries, Ithaca, N. Y.

The Geology and the Copper-Deposits of Bisbee, Arizona, by F. L. Ransome, Washington, D. C.

Geological Relations of the Manganese-Ore Deposits of Georgia, by Thomas L. Watson, Granville, O.

Zinc- and Lead-Deposits of Northern Arkansas, by George I. Adams, Washington, D. C.

Compressed-Air Motors for Gathering Cars in Coal-Mines, by Beverley S. Randolph, Frostburg, Md.

Origin of Pebble-Covered Plains in Desert Regions, by William P. Blake, Tucson, Ariz.

The Box Electric Rock-Drill, by Frank E. Shepard, Denver, Colo.

Biographical Notice of Arthur L. Collins, by B. B. Lawrence, New York City.

The following papers were presented by the author in oral abstract, together with an exhibition of some surveying-instruments used by George Washington :



Magnetic Declination in Northern New York between 1768 and 1900, and its Interpretation in the Surveys, by Verplanck Colvin, Albany, N. Y.\*

Measurement of Base-Lines for Triangulation in Difficult Mountain Districts, by Verplanck Colvin, Albany, N. Y.\*

Wooden Compasses Used by the Colonial Surveyors of New York in 1775, by Verplanck Colvin, Albany, N. Y.\*

President Olcott read a circular letter from the Institute of Great Britain entitled "Ore in sight."

The fourth and last session was held Friday evening, February 20th.

The following papers were presented in oral abstract by the authors, the second being accompanied with numerous lantern-illustrations:

The Ore-Deposits of Sudbury, Ontario, by Charles W. Dickson, New York City.

Operations of the Hudson River Water-Power Company, by Charles E. Parsons, Glens Falls, N. Y.

Captain W. H. Jaques, of Boston, gave a short talk, accompanied by a number of lantern-illustrations, showing appliances used for the utilization of the sun's rays for generating steam.

#### MEMBERS AND ASSOCIATES ELECTED.

The following persons were elected members or associates by postal-ballot of January 2, 1903.

#### MEMBERS.

Huntington Adams,	.	.	.	New York City, N. Y.
Martin Anderson,	.	.	.	Telluride, Colo.
Joseph Argall,	.	.	.	Kamloops, B. C.
Christopher G. Atwater,	.	.	.	New York City, N. Y.
Charles E. A. Ancelle,	.	.	.	Telluride, Colo.
Wm. S. E. Barnardo,	.	.	.	Surbiton, England.
Alberto Bement,	.	.	.	Chicago, Ill.
Charles Biesel,	.	.	.	Spokane, Wash.
Ray Nelson Bishop,	.	.	.	Cleveland, Ohio.
Josiah Bond,	.	.	.	Somerville, N. J.
Philip R. Bradley,	.	.	.	San Francisco, Cal.
Juan Felix Brandes,	.	.	.	Denver, Colo.

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\* Not received in time for publication in this volume.

Wm. A. J. Briggs, . . .	Zeehan, Tasmania.
Alden H. Brown, . . .	Boulder, Colo.
Arthur H. Buck, . . .	Minaca, Mexico.
Irving C. Bull, . . .	New York City, N. Y.
Edward Hyde Burdick, . . .	Victor, Colo.
Alphonse A. Burnand, . . .	Leadville, Colo.
Guillermo Anthony Burr, . . .	Parral, Mexico.
Edwin Arthur Cade, . . .	London, England.
William Campbell, . . .	New York City, N. Y.
Pierre Jules Eugene Chambort, . . .	Paris, France.
Edwin E. Chase, . . .	Denver, Colo.
H. H. Claudet, . . .	Rossland, B. C., Canada.
William Louis Cobb, . . .	San Francisco, Cal.
James Gorman Creveling, Jr., . . .	San Luis Potosi, Mexico.
Uri Balcom Curtis, . . .	Tonopah, Nevada.
William Emanuel Defty, . . .	Phoenix, Arizona.
Louis Charles Deverell, . . .	London, England.
Frederick C. Dobler, . . .	Cornucopia, Oregon.
Willis Gorman Dodd, . . .	San Francisco, Cal.
Almon Emrie, . . .	Easton, Pa.
Ludwig F. Fischer, . . .	San Fernando, Mexico.
Robert Campbell Grimmell, . . .	Mazapil, Mexico.
Edwin Lindley Gormay, . . .	South Australia.
Edward McKim Hagar, . . .	Chicago, Ill.
E. M. Holmes, . . .	Daiquiri, Cuba.
James Ashley Howard, . . .	Pendleton, Oregon.
Masashige Ioka, . . .	Tijima Province, Japan.
Frank A. Jewett, . . .	Hibbing, Minn.
David G. Kerr, . . .	Cordova, Canada.
Edwin Eugene Knapp, . . .	Tonopah, Nevada.
Alfred Church Lane, . . .	Lansing, Mich.
Thomas J. Lawrence, . . .	San Luis Potosi, Mexico.
Lucien Francis Lefevre, . . .	Arras, France.
John Hedges Lidgerwood, . . .	Brooklyn, N. Y.
Jens Lund, . . .	Leonora, Western Australia.
Horace Fletcher Lunt, . . .	Colorado Springs, Colo.
E. K. McCann, . . .	Velardina, Mexico.
William Bernard McCleary, . . .	Tucson, Arizona.
Robert Allen Marr, . . .	Sumpter, Oregon.
George P. Merrill, . . .	Washington, D. C.
George Mesta, . . .	Pittsburg, Pa.
Franz Meyer, . . .	New York City, N. Y.
Frederick H. Morley, . . .	Colorado Springs, Colo.
George Edgar Moulthrop, . . .	Butte, Montana.
Dan Henry Nellis, . . .	Ainsworth, B. C., Canada.
C. T. Orr, . . .	Webb City, Mo.
Thomas Thornton Read, . . .	Laramie, Wyoming.
Frederick W. Reid, . . .	Zeehan, Tasmania.
Paul de Rilly, . . .	New York City, N. Y.
Robert Robinson, . . .	Kalgoorlie, Western Australia.
Will C. Rogers, . . .	Durango, Mexico.
Frederick Wooster Sherman, . . .	Park City, Utah.

Gerald Fitz Gerald Sherman,	. . .	Grass Valley, Cal.
Amasa Delano Sproat, . . .	. . .	San Luis Potosi, Mexico.
Malcolm Reid Stevenson, . . .	. . .	Pittsburg, Pa.
E. G. Thomas, . . . . .	. . .	Independence, Colo.
Edward Thomson, . . . . .	. . .	Pinos, Zacatecas, Mexico.
Francis A. Thomson, . . . .	. . .	Crofton, B. C., Canada.
Arthur R. Turney, . . . . .	. . .	Surrey, England.
Richard R. Van Liew, . . . .	. . .	Globe, Arizona.
Willard Fitzarthur Waller, . .	. . .	Magnet, Tasmania.
Charles E. Watson, . . . . .	. . .	Tonopah, Nevada.
Norval J. E. Walsh, . . . . .	. . .	San Antonio, Texas.
Herbert Joseph Wiswell, . . .	. . .	Mineville, N. Y.
Herman Witteborg, . . . . .	. . .	Idaho Springs, Colo.
Charles E. Wittenoom, . . . .	. . .	London, England.
M. W. H. Woodward, . . . . .	. . .	Silver City, New Mexico.
William Wraith, . . . . .	. . .	Butte, Montana.
Bruce C. Yates, . . . . .	. . .	Lead, South Dakota.

#### ASSOCIATES.

Davenport Brown, . . . . .	. . .	Boston, Mass.
Stanton Watson Forsman, . . .	. . .	Boston, Mass.
James Cuthbert Holderness, . .	. . .	Johannesburg, South Africa.

#### CHANGE OF STATUS.

Reginald Meeks, . . . . .	. . .	Chicago, Ill.
Ivan de Lashmutt, . . . . .	. . .	Santa Rita, New Mexico.

#### EXCURSIONS AND ENTERTAINMENTS.

The session of Tuesday evening was followed by an informal reception in the halls of the same building.

The rooms of the Albany Historical and Art Society were opened to the public March 3, 1898. Their interesting contents were characterized on that occasion by Mayor Van Alstyne, as follows :

"I call your attention to the magnificent collection here on exhibition, representing so amply the double purpose of the Association,—relics in multitudinous variety gathered from periods before the first impress of the white man's foot within the borders of our State down to the present hour. History speaks out from, and art is illustrated in no stinted measure by, every article within these walls."

This description is even more fully deserved at the present time.

Wednesday, February 18th, was spent in an excursion to Schenectady. By courtesy of the General Electric Company, special trolley-cars were furnished for the occasion. After vis-

iting the various departments of their works, luncheon was served by the company at their main office.

In the afternoon the plant of the American Locomotive Company was visited. At present this company is making an extensive enlargement of the plant in the way of foundry and machine-shops, as well as offices and warehouses.

Thursday morning, February 19th, his Excellency, Governor Benjamin B. Odell, Jr., received the members of the Institute and their guests in the Executive-chamber at the State Capitol. Many of the members availed themselves of this opportunity to visit the various departments of the Capitol building, as well as the Senate- and Assembly-rooms.

Friday afternoon, February 20th, as guests of the Albany Chamber of Commerce, the members of the Institute were taken by special cars to the Watervliet Arsenal and witnessed the shrinking of a jacket on a 10-inch gun. The officers in charge of the arsenal took special pains to show the various departments and the machinery used in making heavy guns.

Saturday, February 21st, was devoted to excursions, the visiting members having the choice of two: (1) The Helderberg Cement Company's plant at Howe's Cave, and (2) the Hudson River Water-Power Company's plant at Spier Falls.

By courtesy of the Helderberg Cement Company, a special train was placed at the disposal of the members of the Institute. Upon arrival at the plant, luncheon was served by the company, and afterwards the works were inspected. This is one of the best and largest plants of the kind in the country, and has an annual output of over 600,000 barrels of cement. It is operated entirely by electricity.

The members who did not take this trip went to Spier Falls, where a large dam is being built across the Hudson river by the Hudson River Water-Power Company,\* to supply power for a 50,000-h.p. plant. Through the courtesy of this company a special train conveyed the party to Glens Falls, where sleighs were in waiting to take the visiting members to Spier

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\* E. L. Ashley, President; W. H. Trumbull, Vice-President; E. H. Gay, Treasurer; C. E. Parsons, Chief-Engineer of Construction; G. E. Evans and W. Barclay Parsons, Consulting Engineers. An admirable paper by Mr. C. E. Parsons, describing the construction of the Spier Falls dam, was read at this meeting, and will be published in the *Transactions*.

Falls, 9 miles distant. Upon arriving at the Falls, luncheon was served, the dam and works inspected, and the party returned to Glens Falls in the evening. The inspection of this gigantic piece of engineering-work was supplemented by a banquet at the Rockwell House, Glens Falls.

Besides the excursions and entertainments mentioned above, visiting members were, individually and collectively, the recipients of invitations and courtesies from various institutions, among which may be mentioned the Albany, Aurania, University and Fort Orange clubs.

### MEMBERS AND GUESTS REGISTERED.

The following persons, together with about thirty-five ladies accompanying them, were registered:

E. L. Ashley, . . . . .	Glens Falls, N. Y.
F. E. Bachman, . . . . .	Port Henry, N. Y.
Anson G. Betts, . . . . .	Troy, N. Y.
F. O. Blackwell, . . . . .	Schenectady, N. Y.
R. W. Brock, . . . . .	Kingston, Ont.
W. Howard Brown, . . . . .	Albany, N. Y.
George W. Cabot, . . . . .	New York City, N. Y.
John Cabot, Jr., . . . . .	New York City, N. Y.
J. A. Capp, . . . . .	Schenectady, N. Y.
W. B. Clarke, . . . . .	Schenectady, N. Y.
W. B. Cogswell, . . . . .	Syracuse, N. Y.
Verplanck Colvin, . . . . .	Albany, N. Y.
John Dearborn, . . . . .	Boston, Mass.
W. S. DeCamp, . . . . .	Fulton Chain, N. Y.
C. W. Dickson, . . . . .	New York City, N. Y.
Howard W. DuBois, . . . . .	Philadelphia, Pa.
Theodore Dwight, . . . . .	New York City, N. Y.
W. B. Elmendorf, . . . . .	Albany, N. Y.
C. E. Emmons, . . . . .	Schenectady, N. Y.
H. C. Farnum, . . . . .	Detroit, Mich.
Albert H. Fay, . . . . .	New York City, N. Y.
A. M. Feldman, . . . . .	Albany, N. Y.
Lewis W. Francis, . . . . .	New York City, N. Y.
Hollis French, . . . . .	Boston, Mass.
E. H. Gay, . . . . .	Boston, Mass.
James C. Gray, . . . . .	Boston, Mass.
M. L. Griffin, . . . . .	Mechanicville, N. Y.
A. L. Haasis, . . . . .	Jersey City, N. J.
A. H. Harcourt, . . . . .	Albany, N. Y.
J. B. Hastings, . . . . .	New York City, N. Y.
H. C. Hequemshbourg, . . . . .	Schenectady, N. Y.
J. Hohenemser, . . . . .	Mannheim, Germany.
Foster Hooper, . . . . .	Boston, Mass.

Charles W. Howard, Jr.,	San Francisco, Cal.
A. E. Hoyt,	Albany, N. Y.
Charles Humphrey,	Staten Island, N. Y.
W. S. Hungerford,	Jersey City, N. J.
R. W. Hunt,	Chicago, Ill.
R. G. Hutchins, Jr.,	Columbus, Ohio.
A. O. Ihlseny,	Carthage, Mo.
W. H. Jaques,	Boston, Mass.
W. B. Jones,	Albany, N. Y.
J. N. Judson,	New York City, N. Y.
William Kent,	Passaic, N. J.
Charles Kirchhoff,	New York City, N. Y.
P. Kirkegaard,	Deloro, Ontario.
G. F. Knapp,	Cleveland, Ohio.
J. S. Lane,	Scranton, Pa.
B. B. Lawrence,	New York City, N. Y.
Albert R. Ledoux,	New York City, N. Y.
Louis V. Ledoux,	New York City, N. Y.
S. LeFevre,	Plattsburgh, N. Y.
E. C. Leonard,	Albany, N. Y.
John Lilly,	Lambertville, N. J.
Frank Lyman,	Brooklyn, N. Y.
James F. McElroy,	Albany, N. Y.
J. R. McKee,	Schenectady, N. Y.
A. R. Mackay,	Montreal, Canada.
F. J. H. Merrill,	Albany, N. Y.
Walter Merton,	Frankfort, Germany.
W. G. Miller,	Toronto, Ont.
Richard G. G. Moldenke,	New York City, N. Y.
George S. Morison,	New York City, N. Y.
C. M. Morse,	Buffalo, N. Y.
R. V. Norris,	Wilkes-Barre, Pa.
E. E. Olcott,	New York City, N. Y.
J. Van V. Olcott,	New York City, N. Y.
J. E. Ostrander,	Amherst, Mass.
W. M. Parkin,	Pittsburg, Pa.
C. E. Parsons,	Glens Falls, N. Y.
J. H. Patterson,	Albany, N. Y.
W. A. Pearson,	Schenectady, N. Y.
Frederick J. Platt,	Scranton, Pa.
Herman Poole,	New York City, N. Y.
A. B. Proal,	New York City, N. Y.
J. W. Pullman,	Philadelphia, Pa.
A. L. Reynolds,	New York City, N. Y.
T. A. Rickard,	New York City, N. Y.
J. C. Roberts,	Niagara Falls, N. Y.
T. Robins, Jr.,	New York City, N. Y.
A. L. Rohrer,	Schenectady, N. Y.
L. G. Rowand,	East Orange, N. J.
E. Schroeder,	Cologne, Germany.
H. P. Schuyler,	Schenectady, N. Y.
H. J. Seaman,	Catasauqua, Pa.

R. W. Sherman,	.	.	.	.	.	Utica, N. Y.
H. H. Stoek,	.	.	.	.	.	Scranton, Pa.
G. C. Stone,	.	.	.	.	.	New York City.
E. A. Strong,	.	.	.	.	.	Hartford, Conn.
E. B. Toedt,	.	.	.	.	.	Albany, N. Y.
J. B. Tonking,	.	.	.	.	.	Franklin Furnace, N. J.
W. H. Tonking,	.	.	.	.	.	Dover, N. J.
H. G. Torrey,	.	.	.	.	.	New York City, N. Y.
Enrique Touceda,	.	.	.	.	.	Albany, N. Y.
A. R. Townsend,	.	.	.	.	.	New York City, N. Y.
L. S. Treadwell,	.	.	.	.	.	Albany, N. Y.
W. H. Trumbull,	.	.	.	.	.	Salem, Mass.
Willis G. Tucker,	.	.	.	.	.	Albany, N. Y.
E. A. Uehling,	.	.	.	.	.	Passaic, N. J.
T. E. Van Slyke,	.	.	.	.	.	Albany, N. Y.
Arthur L. Walker,	.	.	.	.	.	New York City, N. Y.
John A. Walker,	.	.	.	.	.	Jersey City, N. J.
Willard P. Ward,	.	.	.	.	.	New York City, N. Y.
George A. White,	.	.	.	.	.	New York City, N. Y.
W. H. Wiley,	.	.	.	.	.	New York City, N. Y.
R. B. Williamson,	.	.	.	.	.	Scranton, Pa.
E. E. Winters,	.	.	.	.	.	Albany, N. Y.
H. J. Wiswell,	.	.	.	.	.	Minerville, N. Y.
F. S. Witherbee,	.	.	.	.	.	Port Henry, N. Y.

Proceedings of the Eighty-Fifth Meeting, New York City,  
October, 1903.

LOCAL COMMITTEES.

*Executive Committee.*—Eben E. Olcott, *Chairman*; Cleveland H. Dodge, *Treasurer*; J. Parke Channing, James Douglas, Benjamin B. Lawrence, T. A. Rickard, Thomas Robins, Jr.

*General Committee.*—W. de L. Benedict, John Bogart, Nicholas Murray Butler, J. Parke Channing, John A. Church, William A. Clark, Frederick Coykendall, Charles F. Cox, F. B. Crocker, A. I. Culver, C. C. Cuyler, Cleveland H. Dodge, James Douglas, Anton Eilers, Daniel Guggenheim, James D. Hague, W. P. Hardenbergh, McDougall Hawkes, Erskine Hewitt, Peter Cooper Hewitt, Alexander C. Humphreys, Frederick R. Hutton, James F. Kemp, Charles Kirchhoff, George F. Kunz, J. Langeloth, Benjamin B. Lawrence, Adolph Lewisohn, John Markle, George W. Maynard, Henry S. Munroe, William H. Nichols, Eben E. Olcott, John F. O'Rourke, S. S. Palmer, Robert Peele, Arthur B. Proal, Jr., James B. Randol, John C. F. Randolph, T. A. Rickard, Thomas Robins, Jr., H. H. Rogers, W. L. Saunders, Horace See, F. L. Sheppard, Charles H. Snow, E. G. Spilsbury, John Stanton, W. A. Stadleman, Joseph Struthers, Robert M. Thompson, John Thomson, William H. Wiley, W. J. Wilgus, H. A. J. Wilkens, David Willcox, Frank S. Witherbee, John McGaw Woodbury.

The headquarters of the Institute during the meeting were at the Murray Hill Hotel.

The opening session was held on Tuesday evening, October 13th, in the Hall of the Board of Education, Park Avenue and 59th Street.

Ex-President Eben E. Olcott, Chairman of the Executive Committee, called the meeting to order, and, with a few felicitous remarks, introduced, as the official representative of Mayor Low, the Hon. McDougall Hawkes, Commissioner of Docks and Ferries of the City of New York, who extended a hearty welcome to the members of the Institute in the name of the city, to which, President Albert R. Ledoux responded.

The following biographical notices\* were presented in oral abstract:—

Biographical Notice of William Earl Dodge, by Dr. James Douglas, New York City.

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\* To be published in full and distributed to members hereafter.



Biographical Notice of George S. Morison, by Dr. Leonard Waldo, New York City.\*

Biographical Notice of J. Peter Lesley, by Benjamin Smith Lyman, Philadelphia, Pa.

Biographical Notice of Theodore D. Rand, by Dr. Thomas M. Drown, South Bethlehem, Pa.

The session concluded with an admirable and most enjoyable exhibition of telephotographic pictures of mountain scenery, by Mr. Dwight L. Elmendorf, of New York City.

The second session was held at Schermerhorn Hall, Columbia University, Wednesday morning, October 16th.

Professor Henry S. Munroe welcomed the members to the University and read in abstract the following papers:—

A Dial Cock for Pulp and Sand, by Henry S. Munroe, New York City.\*

A Laboratory Classifier, by Henry S. Munroe, New York City.\*

The Secretary introduced Mr. Sarat C. Rudra, of Madras Presidency, India, who read the following paper:—

The Mineral Resources of British India, by Sarat C. Rudra, Madras Presidency, India.

During the discussion of this paper, Mr. Rudra gave a very interesting account of gold-mining in Corea.

Luncheon was served at University Hall, and a visit was made to the library and other University buildings, including the departments of Mining, Mechanical Engineering and Metallurgy.

The third session was held at the same place, Wednesday afternoon, October 14th.

The following papers and contributions were presented in printed form by the Secretary, and discussed :

The Garnet-Formations of the Chillagoe Copper-Field, North Queensland, Australia, by George Smith, Sydney, New South Wales.

Observations on Mother Lode Gold-Deposits, California, by William A. Prichard, Kalgoorlie, Western Australia.

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\* Not furnished in time for publication in the present volume.

Some Practical Suggestions Concerning the Genesis of Ore-Deposits, by Max Boehmer, Denver, Colo.

Relative Elimination of Impurities in Bessemerizing Copper-Matte, by W. Randolph Van Liew, Globe, Ariz.

The Metallurgy of the Homestake Ore, by C. W. Merrill, Lead, South Dakota.

The Cyanide-Plant and Practice at the Ymir Mine, West Kootenay, British Columbia, by Edwin C. Holden, New York City.

Discussion of Spurr's paper, "A Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores," by Alex. N. Winchell.

Discussion of Weed's paper, "Ore-Deposits near Igneous Contacts," by W. L. Austin.

Discussion of Lindgren's paper, "Geological Features of the Gold-Production of North America," by Willet G. Miller.

The fourth session was held at the Hall of the American Society of Mechanical Engineers, 12 West 31st St., Thursday morning, October 15th.

The papers selected to be read and discussed at this session pertained mainly to iron and steel metallurgy.

The following paper was read by the author, and was followed by an animated discussion : \*

Tests of Steel for Electric Conductivity, with Special Reference to Conductor Rails, by J. A. Capp, Schenectady, N. Y.

The following paper was read by the author :

The Condition and Action of Carbon in Iron and Steel, by Herbert E. Field, Ansonia, Conn.

The Secretary presented a contribution in discussion thereof by Mr. William Metcalf of Pittsburg, Pa.† The paper was further discussed by Mr. Bradley Stoughton, who illustrated his remarks by a diagrammatic sketch, explaining the "Alloy" theory of the presence of carbon in steel.

The following paper was presented, in the absence of the

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\* The subject of the electric conductivity of rail-steel has been so little studied that the results of experimental work therein would be of great practical value and interest ; therefore it is hoped that the members of the Institute will send to the Secretary statements of whatever experiments they may have carried out in this field, as well as their ideas on the theory of the subject.

† Published in full elsewhere in this volume.

author, by Dr. Leonard Waldo, New York City, who made also some additional observations on the subject:

Electric Steel-Furnace at Gysinge, Sweden, by F. A. Kjellin, Gysinge, Sweden.

The following paper was presented by the Secretary, in the absence of the author:

Hearth-Area, and the Number of Tuyeres in Iron Blast-Furnace Practice, by F. Louis Grammer, Baltimore, Md.

The Secretary presented proposed amendments to the constitution, as follows:

#### I.

It is proposed to amend Rule III. as follows:

Make the first sentences of this Rule read:

"The dues of members and associates shall be \$10 each per annum, payable in advance on the first day of each calendar year; and persons elected at any time during any calendar year shall pay the dues of that year upon election, and shall thereupon be entitled to the publications of the year, as if the said payment had been made at the beginning thereof. And, in addition to the above dues, the Council may at any time prescribe an initiation-fee of \$10, for all candidates thereafter proposed."

#### II.

It is proposed to add to Rule IV. the following:

"The Council may at any time appoint a trustee or trustees, to hold property, real or personal, for the use and benefit of the Institute, upon such terms and conditions as the Council may determine."

#### III.

It is proposed to amend Rule IX. by adding the following:

"The Council is hereby authorized to cause to be created under the statutes of the State of New York, a corporation to hold and administer, for the use and benefit of the Institute, such real or personal property as the Council may, from time to time, transfer to it, such transfer being hereby authorized; said corporation not to be subject to control by the members and associates of the Institute, except through the Council, and in such way as may be provided by the articles of its incorporation, approved by the Council. *Provided*, That the regular business of the Institute, namely, the holding of meetings, and the preparation and distribution of publications, shall remain completely in the control of the Council; and that the funds of the Institute, derived from the dues of members and associates or the sale of publications, shall not be transferred to, or held or controlled by, such corporation, except so far as the Council of the Institute may appropriate money for the necessary expenses of the same—the purpose of this provision being, that the regular business of the Institute, as above defined, shall remain in the hands of the Council, without interference on the part of the said corporation.

"And the Council is hereby authorized to give its assent and co-operation to the proposed plan, based upon the generous offer of Andrew Carnegie, a member of this Institute, to provide a building in the city of New York for the use of

American Engineering Societies; and in case a corporation should be formed to own and administer such building, the Council is authorized to name representation for the Institute in the managing board of said corporation."

#### IV.

It is proposed to amend Rule II., by making the first sentence read as follows :  
"The Institute shall consist of Members, Honorary Members, Associates and Fellows."

Also, by adding the following :

"Fellows shall be life-members, who shall have distinguished themselves in the several professions represented by the Institute, or in original research connected therewith, shall have been connected with the Institute not less than five years, shall have made contributions to its Transactions, and shall have been appointed by the Council in its discretion. And their names shall be placed under a separate heading, or otherwise distinguished, in the List of Members published from time to time.

The Secretary explained that notice of these amendments was given by him, simply in accordance with the Rule, requiring such notice to be given at a meeting preceding the annual meeting; and that the form of the amendments, although approved by a number of members, was not in any way official or final, but must be determined at the annual meeting;—after which, the amendments, if adopted in any form, would be sent to members for final adoption or rejection in that form.

The fifth and final session was held at the American Museum of Natural History, Seventy-eighth St. and Central Park West, on Friday evening, October 16th.

The following paper was presented in abstract and accompanied with experimental illustrations by Mr. George F. Kunz :

The Action of Radium, Roentgen Rays and Ultra-Violet Rays on Minerals, by George F. Kunz and Charles Baskerville, New York City.\*

After the reading of this paper the meeting was adjourned, and the members and guests of the Institute were permitted, through the courtesy of Mr. Morris K. Jesup, President, and Mr. H. C. Bumpus, Director, to inspect the various collections

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\* The full text of this paper was subsequently published in *Science*, Dec. 18, 1903, and therefore will not be reprinted in the *Transactions* of the Institute. Those who desire it may address "*Science*," 41 North Queen Street, Lancaster, Pa.

of the Museum, including especially the Bement collection of gems, recently presented to the Museum by Mr. J. Pierpont Morgan, and not ordinarily open to visitors.

The following papers, not named above, were read by title at the sessions, for subsequent publication and distribution.

Biographical Notice of John F. Blandy, by R. W. Raymond, New York City.

Tombstone and Its Mines, by William P. Blake, Tucson, Arizona.

Notes on Contact-Metamorphic Deposits in the Sierra Nevada Mountains, by H. W. Turner, San Francisco, Cal.

The Copper-Deposits of the Kaibab Plateau, Arizona, by E. P. Jennings, Oak Park, Ill.

Osmosis as a Factor in Ore-Formation, by Halbert Powers Gillette, New York City.

The Yellow-Ocher Deposits of the Cartersville District, Bartow County, Georgia, by Thomas L. Watson, Granville, Ohio.

The Gold-Mining Districts of Central Siberia, by Lewis Blanchard Brown, Irkutsk, Siberia.

The Treadwell Group of Mines, Douglas Island, Alaska, by Robert A. Kinzie, Treadwell City, Alaska.

An Example of the Alteration of Fire-Brick by Furnace-Gases, by Frank Firmstone, Easton, Pa.

Discussion of Grammer's paper, "Flue-Dirt and Top-Pressure in Iron Blast-Furnaces; a Study of the Influences Controlling Them," by Frank Firmstone.

A Proposed Filter-Press Slimes-Plant, by Francis L. Bosqui, San Francisco, California.

A Laboratory Study of the Stages in the Refining of Copper, by H. O. Hofman, C. F. Green and R. B. Yerxa, Boston, Mass.

Hot-Blast Smelting for the Elimination of Arsenic, Antimony, Lead and Zinc from Copper-Mattes, and for the Production of Lead, by S. E. Bretherton, Val Verde, Arizona.

The Assay of Zinc-Box Residues from the Cyanide Process, by Richard W. Lodge, Boston, Mass.

The Refining of the Precipitates Obtained by Means of Zinc in the Cyanide Process of Gold and Silver Extraction, by G. Howell Clevenger, Deadwood, South Dakota.

The Reduction of Lead from Litharge in Preliminary Assays,

and the Advantages of an Oxide Slag, by E. H. Miller, E. J. Hall and M. J. Falk, New York City.

The Power-Plant of the Moctezuma Copper Company at Nacozari, Sonora, Mexico, by John Langton, New York City.

Investigations in Thermal Chemistry Showing Atomic Heat-Valency, by Halbert Powers Gillette, New York City.

The Sperry Vanning-Buddle, by Edwin A. Sperry, Johnston Station, Ga.

Note Concerning an Old Instrument for Finding Distances, Exhibiting the Oldest Known Form of the Transit-Theodolite Principle, by H. D. Hoskold, Buenos Aires, Argentina, South America.

School Laboratory Work: A Free-Milling Gold-Run, by Robert H. Richards and E. E. Bugbee, Mass. Inst. Tech., Boston, Mass.

Copper-Ore and Garnet in Association, by William P. Blake, Tucson, Arizona.

Notes on the Coal- and Iron-Fields of Southeastern Shansi, China, by William H. Shockley, London, England.

Discussion of Prichard's paper, "Observations on Mother Lode Gold-Deposits, California," by H. W. Turner.

Discussion of Jenny's paper, "The Chemistry of Ore-Deposition," by John A. Church.

Discussion of Weed's paper, "Notes on a Section Across the Sierra Madre Occidental of Chihuahua and Sinaloa, Mexico," by G. C. Hewett.

Discussions of Norris's paper, "Water-Hoisting in the Pennsylvania Anthracite Region," by G. A. Burr and C. F. Heathcote.

Discussion of Clarke's paper, "Electrical Apparatus for Coal-Mining," by W. L. Saunders.

#### MEMBERS AND ASSOCIATES ELECTED.

The following persons have been elected members or associates by postal ballot, since the date of the last similar announcement in the proceedings of the Albany meeting of February, 1902.

(In response to Circular No. 2, January 17, 1903.)

## MEMBERS.

Charles Bettendorf, . . . .	Brussels, Belgium.
A. Poyntz Blandy, . . . .	Abingdon, England.
David H. Campbell, . . . .	Denver, Colo.
George Campbell Carson, . . . .	Redding, Cal.
J. P. Casey, . . . .	La Cananea, Mexico.
William O. Crosby, . . . .	Boston, Mass.
Frank M. Drescher, . . . .	Val Verde, Ariz.
Halbert Powers Gillette, . . . .	New York City, N. Y.
Herbert C. Greer, . . . .	Newcastle, Pa.
Thomas Cramer Hopkins, . . . .	Syracuse, N. Y.
Jules Jadot, . . . .	Shanghai, China.
George K. Kimball, Jr., . . . .	Idaho Springs, Colo.
Paul S. King, . . . .	New York City, N. Y.
Chester F. Lee, . . . .	Seattle, Washington.
Francis Church Lincoln, . . . .	Socorro, Mexico.
W. A. Luce, . . . .	Pittsburg, Pa.
Frank Allen Lucy, . . . .	Idaho Springs, Colo.
Edward P. McCarty, . . . .	Minneapolis, Minn.
James F. McClellan, . . . .	New York City, N. Y.
Hood McKay, . . . .	Lykens, Pa.
Willet G. Miller, . . . .	Toronto, Canada.
F. Cushing Moore, . . . .	Wallace, Idaho.
George A. Pratt, . . . .	Minneapolis, Minn.
William Morris Prout, . . . .	Cleveland, So. Africa.
Arthur McBride Ransom, . . . .	Auburn, Alabama.
Will W. Robbins, . . . .	Sumpter, Oregon.
Harry M. Roche, . . . .	Glencoe, Cal.
Francis A. Roepper, . . . .	Empire, Colo.
La Roy Salsich, . . . .	Hibbing, Minn.
Walter Ernest Soest, . . . .	Panuco de Coronado, Mexico.
Joseph William Teale, . . . .	Johannesburg, So. Africa.
John Thomson, . . . .	New York City, N. Y.
John Tonkin, . . . .	Richmond, Va.
Edward Charles Vigeon, . . . .	Anaconda, Mont.
Joseph C. Watkins, . . . .	Galena, Mo.
D'Arcy Wentworth, . . . .	Mt. Morgan, Queensland.

## ASSOCIATE.

Lewis A. Levinsaler, . . . .	San Francisco, Cal.
Franklin Wheaton Smith, . . . .	Minneapolis, Minn.
Harry Thomas Willis, . . . .	New York City, N. Y.

## CHANGE OF STATUS FROM ASSOCIATE TO MEMBER.

Fred Jost, . . . .	San Francisco, Cal.
Walter Scott Lysle, . . . .	Lewiston, Pa.

(In response to Circular No. 3, April 7, 1903.)

## MEMBERS.

Charles Christy Adams, . . .	Mexico City, Mexico.
Harry Clifton Adams, . . .	Philadelphia, Pa.
W. Edward Adams, . . .	Lewiston, Idaho.
A. J. Anderson, . . .	Salt Lake City, Utah.
Frank Seymour Baillie, . . .	Sumpter, Oregon
Frederick W. Bacorn, . . .	Butte, Montana.
Henry J. B. Baird, . . .	Copper Cliff, Canada.
Edwin Tyler Blake, . . .	Mt. Bullion, Cal.
Myrick N. Bolles, . . .	New York City, N. Y.
Durbin H. Bradley, . . .	Philadelphia, Pa.
Lewis Blanchard Brown, . . .	Itat, Siberia.
James Latimer Bruce, . . .	Cripple Creek, Colorado.
Thomas C. Bruce, . . .	Johannesburg, So. Africa.
Chauncey L. Canfield, . . .	San Francisco, Cal.
Thomas Lane Carter, . . .	Johannesburg, So. Africa.
Arthur Wallace Chase, . . .	New York City, N. Y.
Will Lee Clark, . . .	Butte, Montana.
Ira Judson Coe, . . .	Easton, Pa.
Alfred B. Colwell, . . .	Cripple Creek, Colo.
John Cooper, . . .	Concepcion del Oro, Mexico.
Harry Louis Cornish, . . .	Berkeley, Cal.
George F. Corwin, . . .	Catasauqua, Pa.
Robert Parkyn Cosby, . . .	Rochester, Montana.
W. Ray Cox, . . .	Denver, Colorado.
George P. Daniels, . . .	Glenjean, W. Virginia.
Herbert W. Dawson, . . .	Santa Rita, New Mexico.
Arthur Milton Day, . . .	Butte, Montana.
William B. Dennis, . . .	Black Butte, Oregon.
Firmin Vincent Desloge, . . .	Desloge, Missouri.
W. B. Duesler, . . .	Tapalpa, Mexico.
John S. Elliott, . . .	New York City, N. Y.
Herbert C. Farnam, . . .	L'Amable Station, Ontario.
Herbert Edwin Field, . . .	Ansonia, Conn.
Clement B. Finley, . . .	Derby, Conn.
Walter H. Finley, . . .	Atpontley, Tenn.
Alfred Frank, . . .	Butte, Mont.
Harry L. Frank, . . .	Diedrick, Cal.
William W. Garrett, . . .	Great Falls, Montana.
Charles John Garvin, . . .	Denver, Colorado.
J. W. Gates, . . .	Denver, Colorado.
Emerson Gee, . . .	Picacho, Cal.
Henry A. Geisendorfer, . . .	Weimar, Cal.
Walter J. Gorder, . . .	Kimberley, Utah.
William S. Grammer, . . .	Bell Island, New Zealand.
Andrew H. Green, . . .	Detroit, Mich.
William F. Greenard, . . .	Menzies, W. Australia.
Carlton Harper Hand, . . .	Butte, Montana.
Harold H. Harvey, . . .	Oakland, Cal.
Francis T. Havard, . . .	Great Falls, Montana.
H. Lee Huston, . . .	Mokelumni Hill, Cal.



W. F. Jennison, . . . .	Sydney, B. C., Canada.
S. F. Kirkpatrick, . . . .	Kingston, Ontario.
Lochiel M. King, . . . .	Alameda, Colorado.
Fred Shackelford Lack, . . . .	Baker City, Oregon.
Charles Kenneth Leith, . . . .	Madison, Wis.
Chester Munson Lingle, . . . .	Masontown, Pa.
John Lundberg, . . . .	Terry, So. Dakota.
Archibald McIntock, . . . .	Santa Barbara, Mexico.
Winfred S. Marshall, . . . .	Idaho Springs, Colorado.
Edward A. Merton, . . . .	Santiago de Papasquiaro, Mexico.
Frank W. Millard, . . . .	Salt Lake City, Utah.
Enrique de Moulis, . . . .	San Juanito, Central America.
Edmund Naumann, . . . .	Frankfort, Germany.
Carr Baker Neel, . . . .	Sumpter, Oregon.
William Nichol, . . . .	Kimberly, So. Africa.
Peter N. Nissen, . . . .	Prescott, Arizona.
Cleophas C. O'Hara, . . . .	Rapid City, So. Dakota.
Frank W. Oldfield, . . . .	Chinipas, Mexico.
A. B. Patterson, . . . .	Baker City, Oregon.
William E. Pomeroy, . . . .	Sonora, Mexico.
Albert Roberts, . . . .	Montclair, N. J.
Fred C. Roberts, . . . .	Hibbing, Minn.
Robert Parker Roberts, . . . .	Great Falls, Mont.
William Robert Sale, . . . .	Zeehan, Tasmania.
Carl Francis Schader, . . . .	Los Angeles, Cal.
Frank C. Schader, . . . .	Washington, D. C.
Edward H. Scott, . . . .	Callahan, Cal.
Thomas Campbell Scrutton, . . . .	Saranah, Borneo.
Edward W. Sebben, . . . .	Denver, Colo.
George Ernest Silvester, . . . .	Sudbury, Canada.
Charles Alfred Smith, . . . .	San Fernando, Mexico.
Elwynn W. Stebbins, . . . .	Woodside, Cal.
Alvin Arthur Steel, . . . .	Oaxaca, Mexico.
Heath Steele, . . . .	Blue Wing, N. Carolina.
Robert S. Stockton, . . . .	Golden, Colo.
Homer H. Swaney, . . . .	Irondale, Washington.
A. Symons, . . . .	Johannesburg, So. Africa.
Roger Taylor, . . . .	Copper Cliff, Ontario.
C. L. Tingle, . . . .	Georgetown, Colorado.
John N. Tisdale, . . . .	New York City, N. Y.
Clifford C. Thoms, . . . .	Ballarat, Cal.
Howard Throckmorton, . . . .	Denver, Colo.
Louis L. Tower, . . . .	Northport, Washington.
Beverley R. Value, . . . .	Niagara Falls, N. Y.
Fred. Wartenweiler, . . . .	Lead, So. Africa.
Bulkeley Wells, . . . .	Boston, Mass.
Richard Eustis Williams, . . . .	Waihi, Auckland, New Zealand.
Ogden G. Willey, . . . .	Chicago, Ills.
Delos V. A. Williams, . . . .	Oaxaca, Mexico.
James A. Wilson, . . . .	Idaho Springs, Colo.

ASSOCIATES.

George B. Agnew, . . . .	New York City, N. Y.
Sidney M. Bamberger, . . . .	New York City, N. Y.
Frank Rolla Bell, . . . .	Rolla, Mo.
Joseph C. Campbell, . . . .	San Francisco, Cal.
Robert Prewitt Coleman, . . . .	Denver, Colo.
George V. Devinny, . . . .	Edgewater, Colo.
Horace H. Emrich, . . . .	Pueblo, Colo.
W. L. Fleming, . . . .	Golden, Colo.
George C. Foster, . . . .	Golden, Colo.
Lewis Erwin Foster, . . . .	Houghton, Mich.
Marcus Isaac Goldman, . . . .	New York City, N. Y.
Glen Afton Izett, . . . .	Denver, Colo.
Henry Edwin King, . . . .	Colorado Springs, Colo.
Edgar C. Leonard, . . . .	Albany, N. Y.
Charles A. Liddell, . . . .	Golden, Colo.
T. Parker Liddell, . . . .	Golden, Colo.
Lewis Sublet Logan, . . . .	St. Joseph, Mo.
Eugene W. Merwin, . . . .	Golden, Colo.
Frank John Nagel, . . . .	Denver, Colo.
Michael P. O'Brien, . . . .	Milwaukee, Wis.
William B. Rhodes, . . . .	Denver, Colo.
Herbert Arno Roesler, . . . .	Kansas City, Mo.
Johann Heinrich Schmiedell, . . . .	Batavia, Java.
William Arthur Sloan, . . . .	Denver, Colo.
Benjamin E. Tintzman, . . . .	Pittsburg, Pa.
William Field Ward, . . . .	Denver, Colo.
Frank B. Wells, . . . .	Golden, Colo.
Samuel Charles Weil, . . . .	San Francisco, Cal.
Harry J. Wolf, . . . .	Golden, Colo.
William Herbert Landers, . . . .	New York City, N. Y.
Lloyd A. Womble, . . . .	Wardner, Idaho.

(In response to Circular No. 4, June 10, 1903.)

MEMBERS.

Harley J. Armstrong, . . . .	Portland, Oregon.
Emmet D. Boyle, . . . .	Como, Nevada.
Daniel Buckley, . . . .	Crumlynn, Pa.
Leonard H. Carver, . . . .	San Francisco, Cal.
Benjamin H. Case, . . . .	Copperhill, Tenn.
Galen H. Clevenger, . . . .	Deadwood, So. Dakota.
Robert J. Coleman, . . . .	Salt Lake City, Utah.
Frederick N. Cosby, . . . .	Rochester, Mont.
W. Rowland Cox, . . . .	Silverton, Colo.
Charles M. Cross, . . . .	San Francisco, Cal.
Barton Cruikshank, . . . .	San Francisco, Cal.
Arthur E. Davis, . . . .	Guatimapé, Mexico.
C. T. Durell, . . . .	Maiden, Mont.
Newton W. Emmens, . . . .	Pittsburg, Pa.
Henry S. Fleming, . . . .	New York City, N. Y.

John C. Fleschhutz, . . . .	Central City, Colo.
Charles H. Fulton, . . . .	Rapid City, So. Dakota.
Clarence A. Grabill, . . . .	Clifton, Arizona.
Kenneth M. Graham, . . . .	Hamilton, New Zealand.
Samuel M. Greenidge, . . . .	Glen Carbon, Ill.
Harry L. Haehl, . . . .	San Francisco, Cal.
Mathias Hauber, Jr., . . . .	Oaxaca, Mexico.
Kuno B. Heberlin, . . . .	London, England.
Frank C. Hooper, . . . .	North River, N. Y.
Alexander Imhoff, . . . .	Las Vigas, Mexico.
Woolsey McA. Johnson, . . . .	Hartford, Conn.
John B. Keating, . . . .	Winthrop, Cal.
Thomas Kiddie, . . . .	Ladysmith, B. C.
John Laing, . . . .	Rush Run, W. Va.
Charles W. Landrum, . . . .	Carthage, Mo.
Francis B. Lewis, . . . .	Johannesburg, So. Africa.
Elmer J. McCaustland, . . . .	Ithaca, N. Y.
John H. McChrystal, . . . .	Eureka, Utah.
Florentine C. Machalske, . . . .	Brooklyn, N. Y.
James J. Marshall, . . . .	Ansted, W. Va.
Lawrence May, . . . .	St. Louis, Mo.
John F. Morris, . . . .	Gwendoline Mine, Korea.
John T. Morrow, . . . .	Butte, Mont.
Hamilton H. Noyes, . . . .	Johannesburg, South Africa.
Charles Frederick Parker, . . . .	Mapimi, Mexico.
T. Dwight Pickard, . . . .	Fairview, B. C., Canada.
William Poole, . . . .	Port Pirie, So. Australia.
John V. Richards, . . . .	Spokane, Wash.
Frank Scotten, . . . .	Great Falls, Mont.
Edward A. Smith, . . . .	Ixtlan del Rio, Tepic, Mexico.
S. J. Tonkin, . . . .	Parsons, Kansas.
Alexander Veitch, . . . .	Clifton, Arizona.
Harry V. Wallace, . . . .	Altar, Sonora, Mexico.
William L. Whitaker, . . . .	Mitchell, Ind.
William A. Williams, . . . .	San Miguel, Cal.
H. D. Williamson, . . . .	Yokohama, Japan.
H. Clifford Wilmot, . . . .	New York City, N. Y.
James R. Wyun, . . . .	Oxford Furnace, N. J.

## ASSOCIATES.

Emma L. Coryell Ayres, . . . .	Bound Brook, N. J.
Frederick F. Hall, . . . .	Berkeley, Cal.
Jay H. Sabin, . . . .	Joliet, Ill.
Wilson G. Talley, . . . .	Williamsport, Pa.
Sewall Truax, . . . .	New York City, N. Y.

## CHANGE OF STATUS FROM ASSOCIATE TO MEMBER.

A. G. B. Wilbraham, . . . .	Mertola, Portugal.
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(In response to Circular No. 5, August 15, 1903.)

## MEMBERS.

Douglas P. Brown, . . . .	Boscombe, Hants, England.
Willis T. Burns, . . . .	West Falls, Mont.
Norman W. Chapman, . . . .	Seattle, Wash.
Carle D. Clark, . . . .	Eveleth, Minn.
Robert H. Cromwell, . . . .	Santa Barbara, Mexico.
Arthur J. Eveland, . . . .	Metarville, Ga.
William T. Frazier, . . . .	Flat Iron, So. Dak.
Arthur L. Hamilton, . . . .	Butte, Mont.
Willford B. Hoggatt, . . . .	Jualice, Mexico.
Kaichiro Imaizumi, . . . .	Yawata, Japan.
Frank Hayes Keller, . . . .	Great Falls, Mont.
R. C. Kirk, . . . .	Oakland, Cal.
Earl Hamaker Knepper, . . . .	Los Angeles, Cal.
Roger C. Knox, . . . .	Norris, Mont.
Frederic B. Lincoln, . . . .	DuBois, Penna.
Joaquin Maiz, . . . .	Monterrey, Mexico.
Emory T. Miller, . . . .	South Bethlehem, Penna.
Ernest G. Mitchell, . . . .	Santa Barbara, Cal.
Edward W. Moore, . . . .	Aurora, Mo.
Horace G. Nichols, . . . .	Jacksonville, Cal.
Charles J. Norwood, . . . .	Lexington, Ky.
Edward H. Nutter, . . . .	Redding, Cal.
Clyde A. Payne, . . . .	Baker City, Oregon.
Franklin Playter, . . . .	Boston, Mass.
Ralph S. Rainsford, . . . .	New York City, N. Y.
W. V. Richardson, . . . .	Truckee, Cal.
James R. Robertson, . . . .	Sydney, New So. Wales.
Arthur H. A. Robinson, . . . .	Westville, Nova Scotia.
George Branner Rodgers, . . . .	St. Louis, Mo.
Benjamin F. A. Saylor, . . . .	Rome, Ga.
Edmond Norton Skinner, . . . .	Denver, Colo.
Harold C. E. Spence, . . . .	Steeple Rock, New Mexico.
Robert R. Stirling, . . . .	Denver, Colo.
Richard A. Thomas, . . . .	Los Angeles, Cal.
Austin K. Tiernan, . . . .	Salt Lake City, Utah.
Samuel W. Traylor, . . . .	East Orange, N. J.

## ASSOCIATES.

Colby M. Avery, . . . .	Chicago, Ill.
Lavington G. Thompson, . . . .	Launceston, Tasmania.
Harvey Ladew Williams, . . . .	Bristol, Tenn.

## CHANGE OF STATUS FROM ASSOCIATE TO MEMBER.

Lloyd A. Womble, . . . .	Knights, South Africa.
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(In response to Circular No. 6, September 26, 1903.)

## MEMBERS.

John Avery, . . . .	Gippsland, Australia.
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Percy E. Barbour, . . . .	Boston, Mass.
Thomas B. Bassett, . . . .	Santa Barbara, Mexico.
Henry Brooke, . . . .	Hunter, W. Va.
Regis Chauvenet, . . . .	Denver, Colo.
Ernest A. Cleveland, . . . .	Kamloops, B. C., Can.
Leonard C. Crewe, . . . .	Sydney, C. B., Can.
H. Gordon Glore, . . . .	Nogales, Arizona.
Uri B. Grannis, . . . .	Guanajuato, Mexico.
James D. Hague, . . . .	New York City, N. Y.
William B. Haines, . . . .	Santa Rita, N. M.
Lalon F. Hayes, . . . .	San Juancito, C. A.
George C. Hazelet, . . . .	Valdez, Alaska.
W. S. Holloway, . . . .	Los Angeles, Cal.
Lycurgus Lindsay, . . . .	Nogales, Arizona.
Frank E. Marcy, . . . .	New York City, N. Y.
Saxe M. McClintock, . . . .	Inskip, Cal.
Harry L. McNeill, . . . .	Idaho Springs, Colo.
Harry B. Perks, . . . .	Portland, Oregon.
Orville C. Perry, . . . .	Oroville, Cal.
Horace G. Pomeroy, . . . .	Kofa, Arizona.
William A. Pomeroy, . . . .	Cue, Western Australia.
John T. Reid, . . . .	Lovelocks, Nevada.
Clarence C. Sample, . . . .	Aguascalientes, Mexico.
Mortimer A. Sears, . . . .	Baltimore, Md.
David W. Shanks, . . . .	Parral, Mexico.
Roger W. Shapleigh, . . . .	Pueblo, Colo.
Melvorn R. Valentine, . . . .	Victor, Colo.
Henry Stow Young, . . . .	Ban, Borneo.

#### ASSOCIATES.

H. G. Moulton, . . . .	Baker City, Ore.
Frank W. Watson, . . . .	New York City, N. Y.

#### CHANGE OF STATUS FROM ASSOCIATE TO MEMBER.

Charles E. W. Bateson, . . . .	Pilling's Island, Newfoundland.
Franklin W. Smith, . . . .	Bisbee, Arizona.

#### EXCURSIONS AND ENTERTAINMENTS.

Thursday afternoon, October 15th, trips were made to metallurgical and chemical plants in the vicinity of New York City. The following establishments very kindly invited the members to visit and inspect their works: The American Smelting & Refining Co., at Maurer, near Perth Amboy, N. J.; The Nichols Chemical Co., at Laurel Hill, Long Island, N. Y.; The

Balbach Smelting & Refining Co., at Newark, N. J.,\* and the Standard Oil Co. (the Bayonne Refinery), at Bayonne, N. J.

Thursday evening a reception and dance was held at Sherry's, 44th St. and Fifth Ave.

Friday morning, October 16th, through the courtesy of Mr. W. Barclay Parsons, an inspection was made of the Rapid Transit Subway at 42d St. from Madison to Park Ave., and under Park Ave. from 34th to 28th St. A portion of the party visited afterward the deeper tunnel-workings of the subway at 186th St. Through the hospitality of members of the Engineers' Club, luncheon was served at the club-house, 374 Fifth Ave.

Friday afternoon was devoted to a trip by water around Manhattan Island, stopping en route at Rikers Island for an inspection of the plant which is installed there to reclaim submerged land. The boats for this trip were placed at the disposal of the Local Committee by the Department of Docks and Ferries, the Street Cleaning Department, the Cornell Steamboat Co., and the Robins Conveying Belt Co.

Saturday, October 17th, a visit was made to West Point by the steamer "New York" of the Hudson River Day Line, which furnished transportation. Refreshments were served en route in both directions. The party reached West Point at noon and was conveyed by stages to Memorial Hall, where Colonel Albert L. Mills, Superintendent of the Academy, together with the Professors, Instructors, and other officers

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\* The Balbach Smelting & Refining Works, situated in Newark, N. J., on the banks of the Passaic river, is the oldest smelting plant in the United States. The plant is fully equipped with the necessary appliances of a modern metallurgical establishment, and chiefly refines base bullion, copper bullion and silver bullion, but smelts also ores carrying precious metals.

The equipment comprises four 50-ton blast-furnaces, three roasting-furnaces, four softening-furnaces and two refining-furnaces, each of 50 tons capacity. Four 50-ton desilverizing-pots, one reverberatory furnace for antimony-dross, and two reverberatory furnaces for blister-copper.

The silver-department has 11 retort-furnaces for the distillation of silver-zinc scum, four cupel-furnaces, and one electrolytic silver-refinery of a capacity of 40,000 oz. silver per day; one sulphuric-acid silver-refinery of a capacity of 30,000 oz. silver per day, together with the necessary retort- and crucible-furnaces for smelting the fine-silver and fine-gold.

The yearly product of the works is about 18,000,000 oz. of silver, 25,000 oz. of gold, 18,000 tons of electrolytic copper and 40,000 tons of refined lead.

stationed at the post, received the members and guests and conducted them through the various buildings and to other points of interest. During the afternoon the usual review and inspection of the Battalion of Cadets took place, at the conclusion of which a number of the members returned to New York by the steamer, leaving an enthusiastic contingent to enjoy, despite the rain (which began just at that time), a game of foot-ball between the teams of West Point and Harvard, and to return to New York by special car on the West Shore railroad.

#### MEMBERS AND GUESTS REGISTERED.

The following list, which is doubtless incomplete, comprises the names registered at hotel headquarters, omitting those of about 100 ladies accompanying the members.

A. V. Abbott, . . . .	New York City, N. Y.
Thomas J. Adams, . . . .	New York City, N. Y.
Lawrence Addicks, . . . .	Perth Amboy, N. J.
Charles Ancelle, . . . .	Paris, France.
F. E. Bachman, . . . .	Port Henry, N. Y.
Sidney M. Bamberger, . . . .	Salt Lake City, Utah.
John H. Banks, . . . .	New York City, N. Y.
John Birkinbine, . . . .	Philadelphia, Pa. [N. Y.
C. C. Bliss, . . . .	University Heights, New York City,
Wm. H. Boehm, . . . .	New York City, N. Y.
J. H. Bolles, . . . .	Colorado Springs, Colo.
H. T. Bowles, . . . .	New York City, N. Y.
Wm. Braden, . . . .	New York City, N. Y.
Robt. E. Brooke, . . . .	Birdsboro, Pa.
Clarence C. Burger, . . . .	New York City, N. Y.
E. Z. Burns, . . . .	New York City, N. Y.
J. Morison Burnup, . . . .	London, England.
J. A. Capp, . . . .	Schenectady, N. Y.
R. C. Carpenter, . . . .	Ithaca, N. Y.
Wm. J. B. Carries, . . . .	Montclair, N. J.
Ralph W. Carroll, . . . .	New York City, N. Y.
John A. Church, . . . .	New York City, N. Y.
Hiram I. Clark, . . . .	New York City, N. Y.
Josiah H. Clark, . . . .	Paterson, N. J.
J. M. Clark, . . . .	Kanawha Falls, W. Va.
F. Collingwood, . . . .	Elizabeth, N. J.
Robert A. Cook, . . . .	New Brunswick, N. J.
James B. Cooper, . . . .	South Lake Linden, Mich.
Fredk. K. Copeland, . . . .	Chicago, Ill.
Geo. F. Corwin, . . . .	Oxford, N. J.
Torbert Coryell, . . . .	Lambertville, N. J.
W. E. C. Coxe, . . . .	Toledo, O.

Frank H. Crockard,	. . .	Wheeling, W. Va.
Herbert J. Daly,	. . .	Melbourne, Victoria, Aust.
David T. Day,	. . .	Washington, D. C.
Horace S. De Camp,	. . .	Mt. Morris Park, N. Y.
William S. De Camp,	. . .	Fulton Chain, N. Y.
H. V. DeHart,	. . .	Elizabeth, N. J.
J. R. Devereux,	. . .	Aspen, Colo.
J. W. Dougherty,	. . .	Steelton, Pa.
C. D. Drew,	. . .	New York City, N. Y.
E. L. Dufourcq,	. . .	Santa Barbara, Mexico.
E. T. Dumble,	. . .	Houston, Texas.
Theodore Dwight,	. . .	New York City, N. Y.
Wm. S. Edwards,	. . .	New York City, N. Y.
A. Eilers,	. . .	Brooklyn, N. Y.
Karl Eilers,	. . .	New York City, N. Y.
L. V. Emanuel,	. . .	Perth Amboy, N. J.
Chas. Engert,	. . .	Brooklyn, N. Y.
E. F. Eurich,	. . .	Montclair, N. J.
R. H. Eurich,	. . .	Montclair, N. J.
David L. L. Eynon,	. . .	Philadelphia, Pa.
James Eynon,	. . .	Philadelphia, Pa.
Thos. M. Eynon,	. . .	Philadelphia, Pa.
William Eynon,	. . .	New York City, N. Y.
B. F. Fackenthal,	. . .	Riegelsville, Pa.
A. M. Feldman,	. . .	New York City, N. Y.
B. E. Fernow,	. . .	Ithaca, N. Y.
Herbert E. Field,	. . .	Ansonia, Conn.
Stanley G. Flagg, Jr.,	. . .	Philadelphia, Pa.
Lewis W. Francis,	. . .	New York City, N. Y.
Stanton S. Freeman,	. . .	Percyville, Pa.
J. Fritz,	. . .	Bethlehem, Pa.
H. Garlichs,	. . .	Maurer, N. J.
Ware B. Gay,	. . .	Richmond, Va.
Lorenzo M. Gillet,	. . .	New York City, N. Y.
George McM. Godley,	. . .	New York City, N. Y.
A. O. Granger,	. . .	Cartersville, Ga.
R. J. Grant,	. . .	Denver, Colo.
Martin L. Griffin,	. . .	Mechanicsville, N. Y.
H. W. Hardinge,	. . .	Denver, Colo.
Edwin J. Harrington,	. . .	Yonkers, N. Y.
Harry G. Harrington,	. . .	Newark, N. J.
J. E. Haverstick,	. . .	Philadelphia, Pa.
Gus C. Henning,	. . .	New York City, N. Y.
Hiero B. Herr,	. . .	Summit, N. J.
H. D. Hibbard,	. . .	Plainfield, N. J.
Dr. John D. Higgins,	. . .	Ansonia, Conn.
Edwin Chapin Holden,	. . .	New York City, N. Y.
J. A. Holmes,	. . .	St. Louis, Mo.
Geo. S. Humphrey,	. . .	New York City, N. Y.
W. S. Hungerford,	. . .	Arlington, N. J.
Wm. H. Hughes,	. . .	Philadelphia, Pa.
Fred F. Hunt,	. . .	New York City, N. Y.



Edward S. Hutchinson, . . .	Newtown, Pa.
William S. Hutchinson, . . .	Providence, R. I.
W. Spencer Hutchinson, . . .	Boston, Mass.
A. O. Ihlseng, . . .	Carthage, Mo.
A. W. Jackson, . . .	Boston, Mass.
Geo. N. Jeppson, . . .	Worcester, Mass.
Richard M. Jesup, . . .	New York City, N. Y.
Edwin C. Johnston, . . .	New York City, N. Y.
W. J. Johnston, . . .	New York City, N. Y.
F. D. Jones, . . .	Hazleton, Pa.
C. H. Jouet, . . .	Roselle, N. J.
John N. Judson, . . .	New York City, N. Y.
Julius Kahn, . . .	Bloomfield, N. J.
Edward Keller, . . .	Canton, Md.
J. F. Kemp, . . .	New York City, N. Y.
C. W. Kempton, . . .	New York City, N. Y.
John S. Kennedy, . . .	Stanhope, N. J.
Paul S. King, . . .	New York City, N. Y.
P. Kirkegaard, . . .	Deloro, Ontario, Canada.
I. N. Knapp, . . .	Ardmore, Pa.
J. A. Knapp, . . .	New Rochelle, N. Y.
George F. Kunz, . . .	New York City, N. Y.
Edward K. Landis, . . .	Philadelphia, Pa.
J. S. Lane, . . .	Boston, Mass.
Benjamin B. Lawrence, . . .	New York City, N. Y.
Clement Le Boutillier, . . .	Highbridge, N. J.
Robert G. Leckie, . . .	Sudbury, Ontario, Canada.
Albert R. Ledoux, . . .	New York City, N. Y.
J. H. Lee, . . .	Baltimore, Md.
E. Fleming Legle, . . .	New York City, N. Y.
N. Lilienberg, . . .	Philadelphia, Pa.
John Lilley, . . .	Lambertville, N. J.
Stuart Lindsley, . . .	Orange, N. J.
Charles H. Livingstone, . . .	Denver, Colo.
Orleans Longacre, . . .	Spuyten Duyvil, N. Y.
Burdett Loomis, . . .	New York City, N. Y.
A. F. Lucas, . . .	Washington, D. C.
E. J. McCaustland, . . .	Ithaca, N. Y.
Leslie McCreath, . . .	Harrisburg, Pa.
Wm. McMurtrie, . . .	New York City, N. Y.
James MacNaughton, . . .	New York City, N. Y.
P. A. L. Mannheim, . . .	Perth Amboy, N. J.
Henry C. Mather, . . .	New York City, N. Y.
Geo. W. Maynard, . . .	New York City, N. Y.
E. D. Meier, . . .	New York City, N. Y.
George E. Merryweather, . . .	Bridgeport, Conn.
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Richard Moldenke, . . .	Watchung, N. J.
P. R. S. Morison, . . .	Cambridge, Mass.

Henry S. Munroe, . . . .	New York City, N. Y.
E. E. Olcott, . . . .	New York City, N. Y.
Geo. Ormrod, . . . .	Allentown, Pa.
W. B. Page, . . . .	Lambertville, N. J.
Wm. Nelson Page, . . . .	Ansted, W. Va.
Worthington Palmer, . . . .	Albany, N. Y.
I. P. Pardee, . . . .	Hazleton, Pa.
Edward W. Parker, . . . .	Washington, D. C.
S. B. Patterson, . . . .	Allentown, Pa.
C. O. Payne, . . . .	Stamford, Conn.
P. C. Pearson, . . . .	Perth Amboy, N. J.
R. A. F. Penrose, Jr., . . . .	Philadelphia, Pa.
Richard Peters, . . . .	Philadelphia, Pa.
S. H. Pitkin, . . . .	Akron, O.
Stephen M. Pitman, . . . .	Providence, R. I.
Herman Poole, . . . .	New York City, N. Y.
Frank Popplewell, . . . .	Manchester, England.
Mr. Pott, . . . .	Motherwell, Ontario, Canada.
John L. Pultz, . . . .	New York City, N. Y.
Erskine Ramsay, . . . .	Birmingham, Ala.
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E. Richards, . . . .	New York City, N. Y.
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William H. Richmond, . . . .	Scranton, Pa.
T. A. Rickard, . . . .	Staten Island, N. Y.
Charles O. Ripley, . . . .	Newark, N. J.
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S. C. Rudra, . . . .	Mysore, India.
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Harry J. Sheafe, . . . .	Seattle, Wash.
J. M. Sherrerd, . . . .	Easton, Pa.
A. M. Shook, . . . .	Nashville, Tenn.
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# P A P E R S.



# The Ore-Deposits of Sudbury, Ontario.

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(Albany Meeting, February, 1903.)

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## I. THE RELATION OF NICKEL TO PYRRHOTITE.

### *Introduction.*

The Sudbury district is to-day one of the two great sources of nickel in the world. The peculiar geological relations of

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\* This paper was presented by the author as a dissertation for the degree of Ph.D. at Columbia University, New York City.

the ores have long attracted attention, and widespread interest has been aroused by the differences of opinion regarding their origin, whether as igneous segregations or as precipitates from solution. The writer has endeavored to attack this problem with the aid of microscopical and petrographical methods. The field-work of these investigations was carried on during the summers of 1901 and 1902, and the material collected was studied at Columbia University, New York, during the sessions of 1901-'02 and 1902-'03.

As a necessary preliminary, it was thought advisable to investigate carefully the chemical composition and geological relations of pyrrhotite, as well as the mineralogical associations of the Sudbury ores.

### *Minerals of the Sudbury Nickel-Region.*

Pyrrhotite and chalcopyrite, associated with fragments of the enclosing rock, are the predominant minerals of the Sudbury deposits. The nickel-mineral pentlandite, which is the principle source of this metal, is distributed through all the ore-bodies in greater or less amount. Its matrix is almost universally pyrrhotite, though a number of exceptions occur, notably at the Copper Cliff mine.

This mine, which has furnished a number of interesting minerals, contains in the lower levels some massive ore, composed very largely of chalcopyrite intimately associated with pentlandite. Other samples from the same place consist of pyrite and marcasite, also with an intimate mixture of pentlandite.

Millerite is encountered, though rarely. In a sample from the Copper Cliff mine, the writer found small bunches of hair-like crystals of this mineral in the cavities of some radiating pyrite mixed with calcite. It has also been reported in a number of other cases. The millerite is undoubtedly secondary, and was probably derived from pre-existing pentlandite.

Polydymite ( $\text{Ni}_2\text{S}_3$ , with one part of nickel replaced by iron), containing platinum, was described by Clarke & Cattlett.\*

Niccolite, gersdorffite, danaite and arsenopyrite (containing cobalt) occur in several localities in Denison township, though

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\* *Am. J. Sc.*, 3d ser., vol. xxxvii., 1889, p. 372.

the association is not exactly similar to that of the typical deposits.

Pyrite is by no means uncommon, and Walker has described a nickeliferous variety (over 4 per cent. nickel) from the Murray mine. A number of samples of this mineral from the Copper Cliff mine were associated with secondary quartz, calcite and millerite. On analysis, they yielded from a trace to over 3 per cent. Ni, which, however, may be due to an intimate mixture of pentlandite or millerite.

Samples of a compact, whitish or steel-grey mineral, containing a considerable amount of pentlandite, were submitted by the writer to Prof. Penfield, who considers them to be massive marcasite. The analyses conform to the formula  $\text{FeS}_2$ , and show the presence of from 2 to 4 per cent. Ni, probably as pentlandite.

Sperrylite ( $\text{PtAs}_2$ ) was originally found in the gossan at the Vermilion mine. The writer has also isolated it from the unaltered chalcopyrite of the Victoria mine (see p. 11).

Galena occurs rarely in streaks through the pyrrhotite, *e. g.*, at the Mount Nickel mine, Blezard township.\*

Native copper is reported in a few instances in leaves in the associated rocks. Secondary copper-minerals are rare; but bornite is occasionally seen.

Small masses of titaniferous magnetite (with as much as 18 per cent.  $\text{TiO}_2$ ) are at times found.

This list is probably not complete, but represents the most important minerals found by the writer or recorded by others.

S. H. Emmens† describes what he calls several new species of nickel-minerals from Sudbury. But the doubtful purity of his material and the analytical methods employed do not warrant the recognition of these new species.‡

As a preliminary to the discussion of the relation of the nickel to the pyrrhotite and its associates, a brief review of our knowledge of pyrrhotites will be given, since the uncertainty

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\* Specimen kindly furnished by Prof. W. G. Miller, Provincial Geologist of Ontario.

† *J. Am. Ch. Soc.*, vol. xiv, p. 205.

‡ S. L. Penfield, "On Pentlandite from Sudbury, Ont., Can., with Remarks upon Three Supposed New Species from the Same Region," *Am. J. Sc.*, vol. xlv., June, 1893, p. 493.



which prevails regarding the exact chemical composition of pyrrhotites is probably not generally appreciated by students of ore-deposits.

*Pyrrhotites in General.*

For many years the composition of pyrrhotite has presented to chemists and mineralogists a most difficult and interesting problem, to a large extent still unsolved.

Various writers have applied to this mineral formulas which range from  $\text{Fe}_3\text{S}_4$  to  $\text{Fe}_{16}\text{S}_{17}$ . In general terms it is usually expressed as  $\text{Fe}_n\text{S}_{n+1}$ . By some it is regarded simply as FeS with impurities.\* Others consider it as varying molecular mixtures of different sulphides, as  $n\text{FeS} + \text{Fe}_2\text{S}_3$  or  $n\text{FeS} + \text{FeS}_2$ . Bodewig's† and Doelter's‡ formula was  $\text{Fe}_{11}\text{S}_{12}$ .

In other words, "pyrrhotite" is regarded, not as a mineral in the true sense of the term, but as a series of minerals, differing slightly from each other in the relative proportions of their constituents. If this is the correct view, the case is unique. Homologous chemical series are by no means uncommon, but in them there is always a gradual change in the properties of the different members; the extremes being so entirely unlike as hardly to be recognizable components of the same group. There seems, however, to be little, if any, difference between pyrrhotites represented by  $\text{Fe}_3\text{S}_4$  and  $\text{Fe}_{16}\text{S}_{17}$ . It should thus be possible to grade imperceptibly from the monosulphide FeS to the disulphide  $\text{FeS}_2$ , when a pyrrhotite with the composition of pyrite would result. But such a view does not seem reasonable, and is opposed to some of the fundamental laws of chemistry. It seems more reasonable to consider many of the so-called "pyrrhotites" as mixtures of other sulphides with this mineral, and to regard the latter as of a definite composition.

A fact which may have an important bearing on this question is that of the magnetic permeability of different pyrrhotites. In the preparation of various samples, as noted later in this paper, some were observed to be much more magnetic than others, the strength of attraction by a small magnet being, in a

\* Weinschenk, Groth's *Zeit.*, xvii., 499; Lorenz, *Ber. d. chem. Ges.*, 1891, 1501.

† Groth's *Zeit.*, vii., 180.

‡ *Tsch. Mitth. N. F.*, vii., 544. A synopsis of the results obtained by different writers will be found in the *Handbuch der Mineralogie*, Hintze, vol. i., p. 631 et seq.

general way, proportional to the percentage of iron. Further investigation along this line may lead to important results.

It should be borne in mind, however, that lack of care in the preparation of samples for analysis may account for many discrepancies, especially as few pyrrhotites are entirely free from other sulphides in more or less intimate mixture.

The inaccurate methods of analysis, and difficulties of manipulation, will also cause many errors. That the reliability of many analytical methods is open to question is shown by the animated controversies of experts in recent years.\* Errors amounting to 0.75 per cent. in the estimation of sulphur are not uncommon, and in many cases they are possibly larger. The iron-results are also apt to be unreliable.

In a number of determinations on the same sample Habermehl† obtained results in which the sulphur varied from 39.10 to 39.71—a difference of 0.61, and iron from 60.28 to 60.79—a difference of 0.51 per cent. When we consider the theoretical compositions represented by the compounds varying from  $\text{FeS}$  to  $\text{FeS}_2$  it is seen that a comparatively small error would change the formula calculated from the analysis. The calculated composition for different formulas would be as follows:

Formula.	Iron. Per Cent.	Sulphur. Per Cent.	Formula.	Iron. Per Cent.	Sulphur. Per Cent.
$\text{FeS}$ , . .	63.61	36.39	$\text{Fe}_8\text{S}_9$	60.80	39.20
$\text{Fe}_{15}\text{S}_{16}$ , .	62.06	37.94	$\text{Fe}_7\text{S}_8$	60.40	39.60
$\text{Fe}_{11}\text{S}_{12}$ , .	61.60	38.40	$\text{FeS}_2$	46.60	53.40

A few analyses made by prominent workers (see Table I.) will show the wide variation, both in the analytical results and in the recorded specific gravity of the mineral. The latter varies very widely (3.98 to 4.80) and probably indicates the doubtful purity of the sample. Where the total (as in analysis No. 1) is over 101 per cent., any calculations based on the result will be wholly misleading, and it is safe to say that many others recorded, while not indicating such an excess, are equally unreliable for this purpose.

\* Gladding and Lunge, *Journal Am. Chem. Soc.*, vols. xvi., xvii. and xviii.

† *Oberhess. Ges. Nat. u. Heilk.*, 1879, vol. xviii., p. 97.

TABLE I.—*Analyses of Pyrrhotite.*

	1.	2.	3.	4.	5.	6.	7.	8.	9.
Fe.....	68.82	63.15	63.35	62.62	63.41	62.04	61.17	60.83	58.00
S.....	37.36	36.35	35.91	37.38	36.29	38.08	38.83	39.17	42.00
Total.....	101.18	99.50	99.26	100.00	99.70	100.12	100.00	100.00	100.00
Sp. gr.....	3.98	4.62	4.787	4.80	.....	4.497	4.546	4.580	.....
Formula	Nearly FeS.	Nearly FeS.	Nearly FeS. (Fe in excess.)	Nearly FeS.	FeS.	Fe <sub>13</sub> S <sub>14</sub> .	Fe <sub>9</sub> S <sub>10</sub> .	Fe <sub>8</sub> S <sub>9</sub> .	Fe <sub>4</sub> S <sub>5</sub> .

No. 1. Nenntmansdorf. Geinitz, *N. Jahr.*, 1876, 609.

No. 2. Tavetsch. Gutknecht, cited by Kennigott, *N. Jahr.*, 1880, 1, 164.

No. 3. Seeläsgen, Rammelsberg, *Pogg. Ann.*, 1864, cxxi., 368.

No. 4. Brazil, Berthier *Ann. Mines*, 1835, vii., 531.

No. 5. Jeliza, Servia, Losanitsch, *Ber. d. Chem.*, 1892, xxv., 880.

No. 6. Borév, Pálffy, Groth's *Zeit.*, xxx., 184, and xxvii., 101.

No. 7. Bodenmais, Graf Schaffgotsch, *Pogg. Ann.*, 1840, l., 533.

No. 8. Hartzburg, Rammelsberg, *Pogg. Ann.*, 1864, cxxi., 356.

No. 9. Chile, Meunier, *Cosmos*, 1869, 3d series, v., 581.

The above results are taken from C. Hintze's *Hdb. der Min.*, vol. i, 1901, pp. 653 and 654.

Altogether apart, however, from the considerations outlined above, pyrrhotite, as we know it, is not constant in composition; and an adequate explanation of the variations in its composition is one of the problems of the future.

### *The Origin of Pyrrhotite.*

Little is known of the conditions under which pyrrhotite is formed in nature, except that a strongly-reducing atmosphere is necessary. Synthetic reproductions of the mineral, which have not been made under conditions analogous to those of nature, do not aid materially in explaining its origin.

Doelter,\* however, by heating ferrous chloride to 250° C. in an atmosphere of carbonic acid and hydrogen-sulphide gas, produced pyrrhotite much like the natural mineral and having a composition represented by Fe<sub>11</sub>S<sub>12</sub>. He also produced it by the action of H<sub>2</sub>S gas on solutions of ferrous sulphate, carbonate, chloride and silicate. Other methods of preparation do not appear to have any direct bearing on the natural formation of the mineral.

\* Tsch., *Min. u. Pet. Mitt.*, vol. vii., 85, 86.

*Nickel and Cobalt in Pyrrhotite.*

Nickel is universally recognized as an associate of pyrrhotite, and in few cases has it proved, when looked for, to be entirely lacking. Whether the nickel is an essential constituent of the pyrrhotite, replacing part of the iron, or is present in a separate compound, is still a disputed question. It is hoped, however, that the results here presented will help to clear up this matter to a great extent.

The percentage of nickel varies widely in different pyrrhotites. Those occurring in metamorphosed sedimentary and acidic schistose rocks are notably low in nickel, while in those associated with basic igneous rocks, especially of the gabbro family, it may range from a trace to a proportion economically valuable. Cobalt nearly always accompanies the nickel, though usually in very subordinate amount.

Tables II., III. and IV. give the nickel-contents of representative pyrrhotites of these two classes. Those of the first class rarely carry more than 0.5 per cent. of nickel, while those from basic eruptives usually exceed this limit,—some of the Rossland examples being notable exceptions.

TABLE II.—*Percentages of Nickel and Cobalt in Pyrrhotites Generally.*

	Ni.	Co.		Ni.	Co.
<b>ONTARIO.</b>			<b>BRITISH COLUMBIA.</b>		
1. Dalhousie twp. Lanark co.	0.23	trace	1. West Kootenay.....	0.16	0
2. " " " "	0.11	"	2. Kennedy Lake, V. I.....	0.16	trace
3. Angleseatwp. Addington co.	0.16	"	3. Deer Creek, V. I.....	1.70	"
4. Galway twp. Peterboro' co.	0.16	"	4. West Kootenay.....	0.14	"
5. " " " "	0.10	"	5. Crawford Bay.....	0.05	"
6. " " " "	0.05	"	6. Jarvis Island.....	0.28	"
7. Victoria co.....	0.15	"	7. Rossland.....	0.23	"
8. Rainy River District.....	0.13	"	8. ".....	0.13	"
9. District of Nipissing.....	3.30	"	9. Kootenay Lake.....	0.68	"
10. " " " "	2.10	"	<b>SWEDEN, ETC.</b>		
<b>QUEBEC.</b>			1. Klefva.....	1.08	0.07
1. Ottawa co.....	0.13	0	2. ".....	1.50	0.08
2. " " " "	1.68	.....	3. ".....	2.08	0.10
3. Calumet Island.....	1.42	.....	4. Sagmyna.....	0.50	.....
4. " " " "	4.06	0.33		0.80	.....
5. Pontiac co.....	1.50	trace	5. Krageroe.....	1.75	.....
<b>NOVA SCOTIA.</b>			6. Varallo.....	1.20	.....
1. Cape Breton co.....	0.10	"		1.44	.....
<b>NEW BRUNSWICK.</b>			<b>UNITED STATES.</b>		
1. St. Stephen.....	1.82	0.17	1. Gap Mine, Pa.....	1.75	0.10
2. " " " "	trace	.....	2. " " " ".....	1.00	.....
	4.00	.....	3. Anthony's Nose, N. Y.....	0.30	.....

NOTE.—The nickel-mineral gersdorffite occurred in association with some of these pyrrhotites, so that in part the nickel found may be due to an admixture of this mineral.

*Ontario*.—1. Pyrrhotite with pyrite in white translucent quartz and hornblende; 2. Pyrrhotite with pyrite in quartz-mica-diorite (Nos. 1 and 2 occur in dark-grey diorite, which cuts gneiss); 3. Pyrrhotite in dark-grey gneissoid rock; 4. Massive pyrrhotite; 5. Massive pyrrhotite with pyrite, chalcopyrite and quartz; 6. Massive pyrrhotite with chalcopyrite, quartz and feldspar; (other samples from Galway gave from a trace to 0.15 nickel. The pyrrhotite is associated with bands of gneiss, mica-schist and quartzite; i.e., the deposits are “*fahlbands*,” like those of Norway, which are likewise poor in nickel.); 7. Massive pyrrhotite; 8. Massive pyrrhotite with a small amount of quartz from the Huronian schists of English river; 9 and 10. Pyrrhotites with small amounts of chalcopyrite in a gangue of greyish-green diorite. [Pyrrhotites occurring in light- and dark-grey gneissoid rocks from Frontenac county, Ont., Schreiber (Thunder Bay), Darlington Bay (Lake of the Woods), and numerous other localities all contained nickel, but only in traces.]

*Quebec*.—1. Massive pyrrhotite from Eardley township; 2. Pyrrhotite in part massive, in part disseminated through a gangue of quartz, feldspar and hornblende; 3. Small quantity of pyrrhotite, etc., in a quartz-amphibolite; 4. Massive pyrrhotite with a little quartz-gangue (the associated rocks of 3 and 4 are diorites, which cut a series of gneisses and limestones); 5. Massive pyrrhotite.

*Nova Scotia*.—Pyrrhotite in a siliceous gangue from Leitch's Creek.

*New Brunswick*.—Pyrrhotite with a little chalcopyrite and a small amount of quartzose-gangue, or diorite and quartz. (The association is much like that of Sudbury. Various samples from the same locality yield from a trace to 4 per cent. of nickel.)

*British Columbia*.—1. Pyrrhotite in quartz, feldspar and hornblende; 2. Massive pyrrhotite in a gangue of quartz, garnet, calcite and hornblende; 3. Massive pyrrhotite with chalcopyrite in a quartzose-gangue (from the Two Sisters and Crow claim); 4. Pyrrhotite and chalcopyrite in a dark-green rock; 5. Pyrrhotite in association with a small amount of chalcopyrite, graphite, quartz, mica and feldspar; 6. Pyrrhotite with chalcopyrite in a gangue of quartz-green diorite; 7 and 8. Massive pyrrhotite with chalcopyrite in massive eruptive rocks, the ore-bodies lying between gabbros and surrounding porphyries and diabases; 9. Pyrrhotite with pyrite and chalcopyrite in garnet and quartz.

The above Canadian examples are taken from vols. vi. to xi. of the *Reports of the Canadian Geological Survey*.

*Sweden, etc.*—Pyrrhotites in dark eruptive rocks, principally gabbros (authority, Schnabel's *Handbook of Metallurgy*).

*Gap Mine, Pa.*—Pyrrhotite with chalcopyrite, pyrite, etc., in the outer portions of basic igneous rock-masses, which may be metamorphosed to amphibolites.

*Mine at Anthony's Nose, N. Y.*—Pyrrhotite in association with feldspar, pyroxene, hornblende and quartz; the walls being acidic gneisses. (Authority for this and the preceding examples, “The Nickel-Mine at Lancaster Gap,” by J. F. Kemp, *Trans.*, xxiv., 620, 883, October, 1894.)

The relation of the nickel-content to the mode of origin of the pyrrhotites and the enclosing rock will be considered subsequently.

One of the purposes of this investigation has been to test the validity of the view generally held, that in the Sudbury:

pyrrhotites the nickel and cobalt replace the iron isomorphously. Another purpose has been to try to find a definite formula for the sulphide of this district, with the idea of comparing it with similar minerals from other localities.

### *The Sudbury Pyrrhotites.*

The Sudbury nickel-region, as treated in this paper, will be considered, for convenience, as including the deposits of similar nature in the adjoining Algoma district, as well as those in the immediate vicinity of the town of Sudbury.

The ore mined in this region consists chiefly of a mixture of pyrrhotite and chalcopyrite, intimately associated with more or less country-rock. The pyrrhotite carries the nickel, while the chalcopyrite appears to be quite free from that element.\* Table III. gives the nickel and copper in typical samples of the ore of some of the leading Sudbury mines, as taken from the catalogue of the Ontario mineral exhibit at the Buffalo Pan-American Exposition of 1901, and similar data regarding a number of nearly pure pyrrhotites from claims in various parts of Algoma, taken from the reports of the Canadian Survey. These Algoma deposits have the same geological relations—i. e., they are closely associated with an altered gabbro or norite.

The average contents of nickel, copper and cobalt in the ore smelted since 1892 are given in Table IV. The statements of Messrs. Turner and Walker, Table V., show that gold, silver and the metals of the platinum group are constant constituents of the ore, though present only in small quantities.

Sperrylite, the arsenide of platinum ( $\text{PtAs}_2$ ), was first recognized in the decomposed products of chalcopyrite and pyrrhotite at the Vermilion mine. The platinum in the unaltered ore occurs as sperrylite, as proved by the writer.† Mr. T. L. Walker‡ has shown that the sperrylite is associated with the chalcopyrite, rather than the pyrrhotite. Recent investigations seem to confirm this view, as sperrylite has been found associated with copper-minerals in several other places.

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\* Thus, Nos. 3 and 5 in Table III., which are largely chalcopyrite, contain but little nickel.

† *Am. J. Sc.*, February, 1903, 4th ser., xv., 137.

‡ *Am. J. Sc.*, February, 1896, 4th ser., i., 110.

Mr. Turner's estimate, given in columns A and B, Table IV., probably presents fair averages of the general run of the ore. The figures under A may give a good idea of the ore of the Creighton mine, while that from Copper Cliff may run up to 7 per cent. copper and 3 per cent. nickel, according to the particular level from which the ore was taken (copper predominating in the upper, and nickel in the lower levels). Column B is based on the composition of the matte produced, as shown in Table V., column I.

TABLE III.—*Nickel and Cobalt in Sudbury and Algoma Pyrrhotites.*

	Ni. Per Cent.	Cu. Per Cent.		Ni. Per Cent.	Cu. Per Cent.
SUDBURY.			ALGOMA.		
1. Victoria mine.....	3.50	4.50	1. Lorne township.....	1.95	0
2. Copper Cliff mine.....	9.25	0.51	2. " ".....	2.30	0
3. " ".....	1.25	23.78	3. Nairn township.....	1.95	trace.
4. Stobie mine.....	2.99	0.87	4. Drury ".....	2.01	"
5. " ".....	0.64	24.23	5. Denison ".....	1.80	0
6. No. 2 mine.....	4.70	0.88	6. Levack ".....	4.13	.....
7. " ".....	1.87	13.76	7. " ".....	3.00	trace.
8. No. 3 mine. Frood.....	4.85	0.42	8. " ".....	1.96	"
9. No. 4 mine.....	4.33	1.39	9. Morgan ".....	3.30	.....
10. " ".....	4.15	2.49			
11. Creighton mine.....	7.03	1.81			
12. No. 2 extension.....	4.32	0.85			
13. Worthington mine.....	3.00	3.00			

NOTE.—The Sudbury figures are from the catalogue of the mineral exhibit of Ontario at the Pan-American Exposition of 1901, except those of the Worthington mine, which are the result of a number of analyses of average samples, obtained through the kindness of Mr. R. E. Booraem, N. Y. city.

The Algoma figures are taken from reports of the *Geological Survey of Canada*. Most of the samples were massive pyrrhotite, with a small amount of chalcopyrite. The copper per cent. was not reported.

TABLE IV.—*Averages of Smelters' Assays of Sudbury Ores, from 1892 to 1900.*

	1892	1893	1894	1895	1896	1897	1898	1899	1900	A	B
	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	Per ct.
Cu, . .	3.19	2.38	3.14	2.73	2.54	2.86	3.43	.....	1.59	1.50	2.00
Ni, . .	3.36	2.21	2.92	2.67	2.67	2.08	2.28	.....	1.67	3.50	2.50
Co, . .	0.10	0.08	0.07	.....	.....	.....	.....	.....	.....	.....	.....

NOTE.—The figures for the several years are taken from the Sixth Report of the Ontario Bureau of Mines (1896) and *Mineral Industry*, 1900. The returns for 1900 seem too low. Columns A and B, furnished by President A. P. Turner, of the Canadian Copper Co., contain (A) an estimate of certain ores of that company, and (B) a general average based on the composition of the matte produced from a mixture of all its ores.

TABLE V.—*Composition of Sudbury Mattes.*

	I.	II. Per cent.
Copper, . . . .	14 per cent.	25.92
Nickel, . . . .	17 " "	} 48.82
Cobalt, . . . .	0.5 to 0.6 per cent.	
Gold, . . . .	trace	0.000075
Silver, . . . .	0.5 to 1 oz. per ton.	0.001775
Platinum, . . . .	0.25 to 0.5 oz. per ton.	0.000430
Palladium, . . . .	0.25 to 0.5 " " "	present
Iridium, . . . .	.....	0.000056
Osmium, . . . .	.....	0.000056
Rhodium, . . . .	.....	present
Iron, . . . .	.....	2.94
Sulphur, . . . .	.....	22.50

I. Average furnished by President A. P. Turner for mattes of the Canadian Copper Co.

II. Analysis of the matte from the Murray mine, given by Mr. T. L. Walker, *Am. J. Sc.*, February, 1896, 4th ser., i., 112.

To ascertain as accurately as possible the average nickel- and cobalt-contents of the general run of the pyrrhotite from the whole region, a number of analyses of carefully-selected material were made by the writer.

The pyrrhotite was coarsely crushed and the mineral picked out as pure as possible, under a lens when necessary. From the massive varieties good samples were easily obtained; but in other cases the pyrrhotite was so intimately mixed with chalcopyrite and rock that it was very difficult to obtain satisfactory samples, some rock always adhering to the sulphide. The results, given in Table VI., show that the percentage of nickel and cobalt in the pyrrhotite, calculated to the pure mineral, is fairly constant over a wide area, embracing all the principal mines and prospects. The pyrrhotites include both coarse- and fine-grained massive varieties, and those mixed with more or less rock and chalcopyrite. The object was to determine, as far as practicable, only the nickel existing as a component of the pyrrhotite, if it occurred as such. In the case of the coarse-grained varieties, where the nickel-mineral pentlandite is often to be recognized, this was carefully rejected, as far as possible. But the difficulty of this separation accounts for the fact that some of these varieties show less nickel than the fine-grained ones, although the former are usually considerably richer. Had the coarse-grained samples been treated in their



original condition, the results would have been more uniform. Subsequent work proves that a large part, at least, of the nickel is not a constituent of the pyrrhotite. The results of these tests, therefore, represent the nickel which is most intimately associated with pyrrhotite and does not appear in visible particles of pentlandite.

Cobalt is always very subordinate in amount, bearing a ratio to the nickel of between 1:40 and 1:50.

TABLE VI.—*Nickel, Cobalt, etc., in Sudbury Pyrrhotites.*

Location.	Insol. Per Cent.	Cu. Per Cent.	Ni. Per Cent.	Co. Per Cent.	Ni. in Pure Pyr- rhotite. Per Cent.
1. Elsie (a).....	2.00	trace.	2.40	0.06	2.46
2. " (b).....	3.45	"	2.35	0.05	2.44
3. Stobie (a).....	1.50	"	3.00	0.08	3.05
4. " (b).....	4.00	"	2.05	0.05	2.15
5. Frood, No. 3 mine (a).....	0.40	"	2.35	0.05	2.40
6. " " (b).....	5.00	"	2.34	0.06	2.48
7. Mount Nickel.....	2.20	"	3.00	0.07	3.06
8. Copper Cliff, No. 4 mine (a).....	1.10	"	3.24	0.06	3.30
9. " " " 2 " (b).....	5.00	"	3.70	0.08	4.00
10. " " " 5 " (c).....	0.50	"	3.47	0.08	3.50
11. Creighton (a).....	3.25	"	3.84	0.10	4.00
12. " (b).....	0.50	"	2.26	0.06	2.32
13. Gertrude (a).....	5.00	"	3.83	0.11	4.05
14. " (b).....	6.00	"	3.61	0.09	4.00
15. Victoria (a).....	0.50	"	3.36	0.07	3.40
16. " (b).....	0.40	"	3.14	0.08	3.20
17. Levack.....	3.20	"	2.80	.....	2.88
18. North Range.....	4.10	"	2.22	.....	2.32

NOTE.—These analyses, and others by the writer, were made in duplicate or triplicate to insure the greatest possible accuracy.

1. Coarse pyrrhotite with a small amount of chalcopyrite and rock; 2. Compact fine-grained pyrrhotite, with a small amount of rock; 3. Massive fine-grained pyrrhotite; 4. Pyrrhotite and chalcopyrite in diorite; 5. Pure, coarse pyrrhotite; 6. Fine-grained pyrrhotite; 7. Massive pyrrhotite; 8. Coarse pyrrhotite; 9. Massive fine-grained pyrrhotite; 10. Massive fine-grained pyrrhotite; 11. Massive fine-grained pyrrhotite; 12. Coarse pyrrhotite; 13. Massive pyrrhotite; 14. Massive pyrrhotite; 15. Massive fine-grained pyrrhotite; 16. Coarser than No. 15, but with more chalcopyrite; 17. Massive pyrrhotite (Tuff & Stobie's property); 18. Coarse, massive pyrrhotite from Wisner township.

### *The Magnetic Separation of Nickeliferous Pyrrhotite.*

Many experiments have been performed in recent years to effect a commercial magnetic separation of the nickel in pyrrhotite ores; or, for scientific purposes, to determine the con-

dition of the nickel in this mineral. The commercial elimination of the nickel is probably an impossibility, and the scientific problem is only now approaching a solution.

S. H. Emmens\* refers to the work of Habermehl (1879), who effected a separation of European nickeliferous pyrrhotite into magnetic and non-magnetic portions.

T. J. McTighe, in 1890, applied magnetic separation in the treatment of Canadian ores; and in 1892 T. A. Edison, in applying for a U. S. Patent, said:

"I have discovered that where magnetic pyrites, called pyrrhotite, is nickeliferous, as it usually is to a more or less extent, the nickel is not distributed generally throughout the whole body of the pyrrhotite, but certain crystals are pure pyrrhotite or magnetic pyrites, while other crystals have some of the iron replaced by nickel and sometimes by cobalt, and that the crystals containing the nickel and cobalt are considerably less magnetic than the pure pyrrhotite."

Emmens himself made some crude experiments on material from the Gap mine, Pa., and from Sudbury, and obtained two products, the non-magnetic being considerably richer in nickel than the original ore.

Shortly after, David H. Browne† contributed a very valuable article on the same subject. He shows the existence of a rich nickel-mineral in the ores from the Copper Cliff, Evans and Stobie mines, in the Sudbury district, and also that it can be separated by rough-crushing and hand-picking, after first removing the magnetic part. His analyses‡ show that the non-magnetic residue bears a close resemblance to the pentlandite described by Penfield.

Probably the most extensive series of experiments for a commercial separation were those made by Mr. J. N. Judson, of the Wetherill Separating Co., in 1900. The results have never been published; but Mr. Judson has very kindly placed them in the hands of the writer and a partial abstract is here presented.

The material operated on was nearly pure pyrrhotite from Copper Cliff, containing, by analysis, Ni, 3.14; Cu, 0.42; and

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\* *2d Rep. Bureau of Mines, Ont.*, 1892, p. 163. *Jour. Am. Chem. Soc.*, vol. xiv., p. 369.

† *Eng. and M. J.*, December 2, 1893, vol. lvi., p. 565.

‡ Nos. 5 to 8 in Table XI. of this paper.

Fe, 49.78 per cent. The magnetic-separation products, with different current-strengths, and the analysis of each, are given in Table VII.

TABLE VII.—*Magnetic Separation of Pyrrhotite by the Wetherill Separator. First Experiment—Material Crushed to 30-Mesh.*

Product No.	Amperes.	Magnetic Product, Per Cent.	Analyses.					
			Magnetic Portion			Total Percentage of Metal in Magnetic Portion.		
			Ni.	Cu.	Fe.	Ni.	Cu.	Fe.
1	1	90.11	2.46	0.22	53.60	70.58	47.48	97.02
2	15	7.65	10.83	0.78	14.80	26.38	14.37	2.27
3	15	0.78	9.68	2.55	13.25	2.42	4.79	0.21
4	28	0.92	1.76	13.58	20.10	5.87	29.99	0.37
5	tails	0.54	0.70	2.64	11.25	0.13	3.35	0.12
1*	1	90.11	2.46	0.22	53.60	70.58	47.48	97.02
2-5*	...	9.89	9.83	2.21	14.97	29.42	52.52	2.98

The results show that with a current-strength of 1 ampere 90.11 per cent. of the total sample was magnetic, and this contained 2.46 per cent. Ni, or the equivalent of 70.58 per cent. of the total metal in the original pyrrhotite; that is, only 29.42 per cent. of the total nickel was concentrated in the non-magnetic portion (making only two products). The second and third products (15 amperes) were highly nickeliferous; but as they contained less than a third of the total nickel in the ore, the loss in the magnetic portion was very heavy.

In the next experiment, the first magnetic product at 1 ampere (amounting to 90.11 per cent. of the total), containing Ni, 2.46; Cu, 0.22; and Fe, 53.60 per cent., was again treated, using a weaker current. The results are shown in Table VIII.

TABLE VIII.—*Magnetic-Separation Product No. 1 of Table VII. Second Experiment (No. 30-Mesh).*

Product No.	Amperes.	1. Magnetic Portion of Part Treated.	2. Magnetic Portion of Total.	Analyses.		
				Ni. Per Cent.	Cu. Per Cent.	Fe. Per Cent.
1	$\frac{2}{15}$	33.61	30.28	1.93	0.08	56.65
2	$\frac{3}{15}$	54.15	48.79	2.08	0.13	55.60
3	$\frac{4}{15}$	4.64	4.18	3.46	0.18	53.00
4	$\frac{5}{15}$	1.79	1.61	3.70	0.51	41.25
5	1	2.35	2.12	3.95	0.90	23.80
6	tails	3.47	3.13	10.60	2.44	20.20

\* Making only two products—i.e., magnetic and non-magnetic—at one ampere.

Column 1 gives the percentage of the part treated, which was magnetic with the different current-strengths indicated; column 2, the percentage of the total sample which was magnetic under the same conditions (*e.g.*,  $90.11 \times 33.61 = 30.28$ ). The magnetic portion at  $\frac{2}{16}$  amperes (about  $\frac{1}{3}$  of the total) still contained nearly 2 per cent. Ni; so the loss was still very great, while the non-magnetic concentrate was not very greatly enriched.

All the products of these two experiments were then mixed, crushed to pass through 60-mesh and treated as shown in Table IX.

TABLE IX.—*Magnetic Separation of Mixed Sample Crushed to Pass No. 60-Mesh. Third Experiment.*

Product No.	Amperes.	Magnetic Portion. Percent. of Total.	Analyses.		
			Ni.	Cu.	Fe.
1	$\frac{1}{16}$	44.26	1.69	0.13	55.50
2	$\frac{1}{8}$	41.96	2.16	0.18	54.80
3	$\frac{1}{4}$	0.37	2.16	0.26	50.30
4	$\frac{1}{2}$	0.37	4.19	0.70	32.50
5	1	0.49	5.29	0.93	24.55
6	2	1.13	8.96	0.97	20.85
7	3	1.38	13.55	0.76	21.55
8	4	3.95	10.29	0.46	22.35
9	5	4.19	12.71	0.60	19.60
10	6	0.89	4.17	3.18	12.65
11	28	0.77	1.53	14.36	19.65
12	tails	0.52	0.38	1.64	8.25

At  $\frac{1}{8}$  ampere, 44.26 per cent. of the sample was magnetic, and product contained 1.69 per cent. Ni. At  $\frac{1}{4}$  ampere, 86.22 per cent. was magnetic, and the product contained 1.92 per cent Ni. If the remainder, 13.78 per cent., were considered as the non-magnetic concentrate, we would have a comparatively rich nickel-ore, but the losses in the magnetic portion are so great that a commercial separation by this method is out of the question. All the other experiments led to the same conclusion. Intermediate products, rich in nickel, were easily obtained; but in no case was the nickel in the magnetic portion reduced to such an extent that it could be economically rejected.

Mr. D. P. Shuler, Sudbury, Ont., has recently (1902) taken out patents for a process whereby he proposes to eliminate the

copper from the nickel-iron portion by magnetic concentration, and subsequently to convert the latter, after roasting, into a nickeliferous pig-iron. The copper-nickel concentrate will, of course, be treated separately for its metallic contents. This seems to be the most fruitful field for investigation now open in this connection, and about the only method of treatment which promises to yield results capable of industrial application.

*The Nickel-Bearing Mineral.*\*—Prof. S. L. Penfield† first definitely proved the existence of pentlandite in the ores from the Sudbury nickel-copper mines. Later, Mr. David H. Browne‡ showed that pentlandite was the principal nickel-bearing mineral. On the assumption that the ores were the result of a magmatic segregation from an original fused magma, he tried to show that the coarser the grain of the pyrrhotite, and the deeper it lies below the surface, the more nickel exists as pentlandite. On the other hand, the finer-grained the ore, and the more rapidly it has cooled from the fused state (*i.e.*, the nearer it is to the surface), the more nickel exists as an element replacing the iron in the pyrrhotite. As will appear later in this paper, however, this relation does not hold good.

In order to determine as nearly as possible how much of the nickel occurs as a separate mineral, and how much, if any, replaces iron, and also to ascertain the composition of the nickel-mineral, several series of experiments were made by the writer.

*First Series of Experiments.*—A number of representative samples of pyrrhotite were ground to pass through 100-mesh and the non-magnetic portion was removed as completely as possible by repeated treatment with a small horse-shoe magnet. The nickel-contents of the original samples are given in column I., and those of the magnetic concentrate in column II., Table X. The nickel is seen to have been materially reduced, and the results seemed to indicate a pretty constant quantity remaining with the magnetic part. Further experiments, however, showed that this was purely an accidental relation.

*Second Series of Experiments.*—The original samples were coarsely crushed and the magnetic portion was sized between

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\* A preliminary note on this subject was published in the *Eng. and Min. Jour.*, May 10, 1902, by the writer.

† *Am. Jour. Sc.*, vol. xlv., June, 1893, p. 493.

‡ *Eng and Min. Jour.*, Dec. 2, 1893, and *Sch. of Mines Quart.*, vol. xvi., p. 297.

40- and 60-mesh, then freed as well as possible from non-magnetic material, crushed between 60- and 80-mesh and again concentrated. By successive treatments the mineral was finally reduced to a fine powder. The ultimate product was then assayed for nickel. As shown in column III., Table X., the nickel was much reduced, but not entirely eliminated.

*Third Series of Experiments.*—To see if it were possible still further to reduce the nickel-contents, a number of samples were very carefully prepared. They were coarsely crushed and the purest mineral selected. This was crushed to pass through 10- on 20-mesh, and the finer material was rejected. All the non-magnetic portion was eliminated and the concentrate was then crushed to 20- on 40-mesh, the finer part being again rejected. The operations were repeated until the ore was finally ground in an agate mortar, the non-magnetic part being very carefully removed each time. The nickel in the final concentrate is given in column IV., Table X.

TABLE X.—*Experiments in the Magnetic Concentration of Nickeliferous Pyrrhotite.*

Location.	I. Ni.*	II. Ni.	III. Ni.	IV. Ni.	
1. Elsie mine.....	2.44	2.22	0.98	.....	Fine-grained pyrrhotite.
2. Stobie mine.....	3.05	2.14	0.68	.....	“ “
3. Frood mine.....	2.40	2.07	1.05	0.65	Coarse-grained pyrrhotite.
4. Mount Nickel mine.....	3.06	2.14	0.75	0.70	Medium-grained “
5. Copper Cliff, No. 2 mine.	4.00	2.00	0.70	.....	Coarse-grained “
6. “ “ No. 4 “	3.30	2.32	0.83	.....	“ “ “
7. Creighton mine (a).....	2.32	2.25	1.20	0.70	“ “ “
“ “ (b).....	4.15	.....	.....	0.45	Fine “ “
8. Gertrude mine.....	4.00	2.30	1.10	.....	Massive pyrrhotite.
9. Victoria mine.....	3.40	2.46	0.80	.....	Fine-grained pyrrhotite.

The results show conclusively that the nickel present does not replace part of the iron in the pyrrhotite, but exists as a separate mineral. The fact that all the nickel could not be eliminated by the methods used does not indicate that even the amount that remained was an essential part of the pyrrhotite, as several factors enter which render its complete removal practically impossible. In the first place, the nickel-mineral is very intimately associated with the magnetic pyrites; and

\* Nickel with a trace of cobalt.

even a minute adhering fragment of the latter will cause it to be carried over with the magnetic portion. It is also to be noted that the nickel-mineral itself is slightly magnetic, and in the form of a powder is attracted by even a small magnet.

As a result of these experiments I feel justified in saying that all the nickel, in the Sudbury ores at least, occurs as a separate mineral, and that in this district there does not exist a true nickeliferous pyrrhotite, in the sense that the nickel isomorphously replaces part of the iron in that mineral.

Experiments on Swedish and Norwegian ores of a similar nature show that a large part of the nickel can be eliminated by magnetic concentration; and taking these results, in connection with the foregoing, it is pertinent to ask: "Is there such a thing as a true nickeliferous pyrrhotite?"

The results show that even with the most careful treatment, a commercial separation of the nickel by magnetic concentration is prohibited by the considerable loss of nickel involved.

*The Non-Magnetic Residue.\**—Analyses of the non-magnetic residues, roughly freed from impurities, were made to get an idea of their compositions.

The small amount of copper was estimated as chalcopyrite, and the necessary amounts of iron and sulphur deducted. The ratios of Fe : Ni : S (taken as 1) were then calculated, and the results are given in Table XI.

TABLE XI.—*Analyses of the Non-Magnetic Residue from Pyrrhotites.†*

Location.	Cu. Per Cent.	Total Fe. Per Cent.	Total S. Per Cent.	Ni. Per Cent.	Co. Per Cent.	Fe.‡ Per Cent.	S.‡ Per Cent.	Ratios. (S. 1.00.)			
								S.	Ni.	Co.	Fe.
1. Victoria mine.....	0.95	28.50	31.92	31.65	0.65	27.66	30.96	1.00	0.570		0.514
2. Frood mine.....	0.84	29.36	32.85	32.25	0.84	28.65	32.04	1.00	0.565		0.512
3. Creighton mine.....	1.20	29.50	34.25	33.00		28.45	32.80	1.00	0.550		0.50
4. General sample.....	0.80	27.11	30.30	29.71	0.60	26.40	29.46	1.00	0.565		0.515
5. Copper Cliff mine.‡	0	.....	.....	35.05		29.80	34.35	1.00	0.56		0.50
6. " " " " " "	0	.....	.....	35.00		30.30	33.50	1.00	0.571		0.52
7. Evans mine.....	0	.....	.....	34.90		29.60	33.35	1.00	0.571		0.51
8. Stobie mine.....	0	.....	.....	34.70		29.90	33.90	1.00	0.56		0.505

\* The residue is spoken of for convenience as being non-magnetic, although in reality slightly magnetic.

† For the sake of uniformity the following revised atomic weights are used throughout: Ni, 58.69; Co, 58.99; Fe, 56.02; S, 32.07.

‡ After deducting the necessary amounts to form chalcopyrite with the copper present.

§ Analyses, by D. H. Browne, Nos. 5 and 8 contained some pyrrhotite; Nos. 6 and 7 were purer. *Eng. and Min. Jour.*, December 2, 1893.

The striking uniformity of the ratios obtained at once indicated the similarity of the mineral in all these cases; and further work on carefully-purified material confirms this view.

The pyrrhotite from the Creighton, Worthington and Frood mines shows considerable amounts of the non-magnetic mineral, in pieces up to  $\frac{3}{4}$  in. in diam., fairly pure, and specimens from these localities were chosen as affording the best material to work on.

The ore was first roughly crushed and the pyrrhotite removed by a magnet. The larger non-magnetic fragments were taken and picked over under a lens, carefully discarding any that was even suspected of containing pyrrhotite.

The samples were then crushed successively to pass through No. 20-, No. 40-, No. 60-, No. 80-, and No. 100-mesh, each time discarding the fines and going over them with a magnet, examining under a lens, and removing all foreign material till they were too fine to treat in this way. They were finally ground in an agate mortar and again gone over with a weak magnet, which attracted the mineral but little and served to remove any traces of pyrrhotite that might remain. In this way the samples were obtained as pure as it is practically possible to get them, and they were ready for analysis; the results are shown in Table XII.

TABLE XII.—*Analyses of Nickel-Bearing Mineral.\**

Location.	Cu.	Ni.	Co.	Fe.	S.	Ratios. (S.=1.00.)		
						S.	Ni.+Co.	Fe.
1. Creighton .....	0	34.82	0.84	30.00	32.90	1.00	0.589	0.518
2. Worthington....	0	33.70	0.78	29.17	32.30	1.00	0.583	0.517
3. Frood.....	0	34.98	0.85	30.04	33.30	1.00	0.588	0.513
4. Copper Cliff †...	0	34.23	0.85	30.25	33.42	1.00	0.573	0.518

The composition of the non-magnetic residue at once suggests pentlandite, and as such Penfield described the sample he examined (No. 4 in Table XII.).

\* The small amount of residue consisted of siliceous gangue. These analyses were made in the laboratory of the School of Mining, Kingston, Ont., which was kindly placed at the writer's disposal.

† Penfield, *Am. J. Sc.*, vol. xlv., 1893, p. 493.



The pentlandite from Lillehammer, Norway, corresponds very closely to it in physical properties, but carries a lower percentage of nickel (20 to 22 per cent.).

The Sudbury mineral occurs, in part, finely disseminated through the pyrrhotite, so that it cannot be detected, even with the aid of a lens, and partly in larger grains and segregations, at times an inch in diam. Usually it possesses the characteristic platy structure and octahedral parting, though this is at times obscured. On fresh fracture, the mineral has a light steel-grey or silver-white color, which soon changes on exposure to the peculiar and characteristic light bronze-yellow. When fresh and massive it is with difficulty to be distinguished from the fresh pyrrhotite, but on a slight superficial oxidation the difference becomes quite pronounced.

The Lillehammer pentlandite has a ratio of  $\text{FeS} : \text{NiS}$  about 2 : 1, and its formula is given as  $2\text{FeS} + \text{NiS}$ . Penfield\* gives the ratio of the Sudbury mineral as  $(\text{Fe} + \text{Ni}) : \text{S} = 1 : 1$ , or  $(\text{Fe} + \text{Ni}) \text{S}$ .† By a reference to Table XII. it will be seen that the ratio  $(\text{Fe} + \text{Ni}) : \text{S}$  is (1) 11.07 : 10; (2) 11 : 10; (3) 11.03 : 10; and (4) 10.91 : 10; that is, there is an excess of the metallic constituents over the amount required by the formula  $(\text{Fe} + \text{Ni})\text{S}$ . The same excess is indicated in the analyses of less pure material given in Table XI. This relation is constant and cannot be regarded as accidental, especially since the analyses of the purest samples from different localities show the ratio to be constant at 11 : 10. Just what this peculiarity signifies it is difficult to say; but it seems to indicate that the structure of the pentlandite molecule is more complicated than that represented by  $(\text{Fe} + \text{Ni})\text{S}$ , even though it is advantageous to represent minerals by the simplest possible formulas.

An excess of sulphur over the metal is by no means uncommon in minerals (*e.g.*, in pyrrhotite, polydymite, beyrichite, etc.); but an excess of metallic contents seems unique.

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\* *Loc. cit.* The analysis given by Prof. Penfield, in this paper, agrees almost exactly with those made by the writer, but by a slight error in calculation, his analysis worked out exactly to the formula  $(\text{Fe} + \text{Ni}) \text{S}$ , instead of showing an excess of metallic constituents, as is actually the case; that is the ratio of  $(\text{Fe} + \text{Ni}) : \text{S}$  in the material examined by Prof. Penfield is almost 11 : 10, instead of 1 : 1 as given.

† Nickel and cobalt being taken together.

Such a formula as  $(\text{Fe} + \text{Ni})_{11}\text{S}_{10}$  is not convenient; but it appears to be the only means of expressing the results of analysis.

Another interesting feature is the ratio of nickel to iron, which is quite constant also, nearly 11 : 10, while that of nickel to cobalt is about 42 : 1.

An intimate admixture of some sub-sulphide with what might be regarded as normal pentlandite  $[(\text{Ni} + \text{Fe})\text{S}]$  is, of course, a possibility; but if this is the case it must be in a definite and constant ratio, as the mineral shows the same phenomenon throughout the whole district.

But this is a mere speculation. There is no direct evidence on which to base the assumption, and at present it seems most reasonable to imagine a peculiar molecular arrangement of the atoms of the mineral itself. As the matter now stands, however, the relations are unexplained, and must be the subject of further investigation before a satisfactory conclusion is reached.

#### *The Formula of the Pyrrhotite.*

A large number of the magnetically-separated samples were analyzed, in an attempt to determine the formula of the pyrrhotite, and with a fair amount of success. A number of factors had to be taken into consideration, and tended to render the results in some cases rather unsatisfactory, so that absolute uniformity was not attained.

The nickel present was calculated to pentlandite on the basis of the ratios given above, and the necessary amounts of iron and sulphur were deducted. As there was a persistent, though varying amount of magnetite in all the samples, it was necessary to determine this. The estimation of the oxide ( $\text{Fe}_3\text{O}_4$ ) in the presence of the magnetic sulphide presented considerable difficulty, and this is probably the main cause of the discrepancies.

After a number of experiments, it was found that by treating the sample with a dilute (10 per cent.) solution of nitric acid, the pyrrhotite could be largely removed, while the magnetite was but little affected. The separated sulphur was removed by means of bromine and carbon-bisulphide; and after several treatments the residue of magnetite was obtained pure, and the iron was estimated by titration. The nature of the various

operations involves the possibility of some loss, especially as the amount of magnetite is comparatively small; but, on the whole, the method answered very well. The great majority of the analyses indicated that the pyrrhotite could be represented by the formula  $\text{Fe}_8\text{S}_9$ , while two or three worked out to  $\text{Fe}_7\text{S}_8$  and  $\text{Fe}_9\text{S}_{10}$ ; so that the former can be regarded as the most probable for the pyrrhotite from the Sudbury district.

For purposes of comparison, a number of pyrrhotites, as pure as it was possible to obtain them, from various localities, were further purified by magnetic concentration and analyzed. One from Rossland, B. C., conformed to  $\text{Fe}_8\text{S}_9$ , one from Anthony's Nose, N. Y., to  $\text{Fe}_7\text{S}_8$ , as did also a sample from Enterprise, Ont.\*

As absolute uniformity, even in the material from a limited district, was not obtained, it would not be wise to draw general conclusions, or attempt to make any general applications, without an exhaustive study of more representative material.

It seems, however, a justifiable conclusion from the work so far, that the pyrrhotite from the nickel-region of Sudbury is of fairly uniform composition, and is best represented by the formula  $\text{Fe}_8\text{S}_9$ .

#### *Summary.*

In recapitulation, the main facts may be summed up briefly as follows:

1. Our present conception of the constitution of pyrrhotite is very unsatisfactory. The nature and associations of the mineral are such that analyses are often misleading, and a great deal of careful, systematic work is necessary as a basis for generalizations of any value.

2. The conditions under which pyrrhotite is formed in nature are still very little understood.

3. Nickel is universally associated with pyrrhotite in greater or less amount, though in but few instances is it present in quantities of economic importance.

4. The Sudbury pyrrhotites are very uniform over large areas, as to metallic contents and associated minerals.

5. Aside from the copper- and nickel-minerals, the others

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\* This pyrrhotite is associated with a remarkable deposit of molybdenite, which has recently been exploited.

are of small economic importance, except, perhaps, sperrylite, which is furnishing an increasing amount of the world's supply of platinum.

6. The magnetic separation of the nickel from pyrrhotite is out of the question commercially.

7. The nickel occurs in the pyrrhotite as the so-called pentlandite, and not as an essential constituent of this mineral.

8. Nearly all the pentlandite can be separated from the pyrrhotite by magnetic methods; but peculiar physical conditions seem to render its absolute elimination an impossibility.

9. The pentlandite does not conform to the generally accepted formula  $(\text{Fe} + \text{Ni})\text{S}$ . The metallic constituents are always in excess of the sulphur by a constant ratio of 11:10. But, as yet, no satisfactory explanation of this phenomenon can be advanced.

10. The pyrrhotite from Sudbury can best be represented by the formula  $\text{Fe}_9\text{S}_{10}$ .

11. Pyrrhotite and magnetite can be separated by a 10 per cent. solution of nitric acid with fairly satisfactory results.

## II. GENESIS OF THE SUDBURY ORES.

### *General Considerations.*

For convenience, pyrrhotites may be divided into two main classes, according to their geological relations.

1. Those of the first class occur, often with more or less pyrite and chalcopyrite, as lenses in acidic gneisses and schists. These lenticular masses generally conform to the foliation of the schists, and are often repeated and connected by leaner zones, like fahlbands. Such deposits are of world-wide distribution; and, wherever found, generally carry pyrrhotite low in nickel, seldom containing more than 0.5 and usually less than 0.25 per cent.

2. The second class also is widely distributed. The deposits are likewise lenticular in shape, but they are associated with basic igneous rocks, usually of the gabbro type, or their metamorphic equivalents. Another characteristic feature is, that while they lie well within the limits of the eruptive rock, they generally occur at or near its contact with other rocks, such as granite, mica-schist, porphyry or diabase. The pyrrhotite is

always associated with more or less chalcopyrite and pyrite, and at times with the rarer minerals, pentlandite, sperrylite and gersdorffite. The characteristic minerals of the basic rock are also intimately mixed through the prevailing sulphides. Nickel is almost invariably present, at times reaching the amount of 10 per cent. Generally, however, from 2 to 4 per cent., as at Sudbury, would be a fair average. In Norway and Sweden the average is lower.

It should be added that the above general statements, as to the nickel-contents of the two classes of pyrrhotite-deposits, is subject to some important exceptions.

3. Outside of these two classes, pyrrhotite is found at times in considerable quantity in true fissure-veins; but this occurrence is relatively unimportant.

Types of the first class, as represented by the Phillips mine, at Anthony's Nose, N. Y., the Ducktown, Tenn., deposits, and numerous *fahlbands* in various parts of Ontario, Norway and Sweden, are believed to be replacement-deposits along crushed zones.

To the second class of deposits an igneous origin has been attributed in recent years by some of the ablest workers in this field of geology. The sulphides are regarded as original rock constituents, and it is thought that, being among the first minerals to crystallize in a cooling rock-magma, they became concentrated in their present position before the rock solidified.

It is well established that rock-magmas—especially the more basic—tend to divide into fractional parts of varying acidity. What forces produce this "magmatic differentiation" are not clearly understood.

There seems to be excellent evidence to prove that many deposits of magnetite (always titaniferous), chromite and corundum have originated in this way. That is, they are simply excessively basic developments of magmas, in which the mineral in question is a normal, or common, accessory constituent.

These deposits, however, consist of oxidized compounds; and their segregation presents fewer theoretical difficulties than does that of the sulphides.

*Value of the Classification.*—Quite apart, however, from theories of origin, the distinction above made as to geological relations is of the utmost importance from an economic standpoint.

This view of the relative value of the pyrrhotites in acid and basic rocks, as nickel-ores, is strongly urged by Profs. Adams and Kemp in valuable papers which appeared about the same time.\*

*The Sudbury Pyrrhotite-Deposits.*

In studying the origin of pyrrhotite of the second class, particular attention has been paid to the Sudbury deposits. Reference will, however, be made to similar deposits elsewhere when necessary.

Before proceeding with the consideration of the origin of the ores, it will be necessary to give a brief account of the geology of the deposits, especially as the ideas formerly held have been recently much modified. The Canadian Geological Survey has lately paid particular attention to the district, and it is now recognized that its relations are much more complex than was at first thought. In his preliminary report on the district, Dr. A. E. Barlow,† who has had charge of the work, gave some general ideas of the nature of the rocks and the extent of the ore-bodies. But the field-work was not completed at the time, and many new facts have been collected during the past season; hence, his final views on the subject will not be presented until the full report is issued. Meanwhile, the question must be considered with the understanding that some of the previously-presented conceptions are subject to revision.

The deposits are prevailingly lenticular, pinching out in both directions and conforming to the general strike of the enclosing Huronian strata. The ore always occurs in, and contains fragments of, a basic and altered eruptive of the gabbro type.

The ore-bodies may occur either well within this eruptive or at its contact with the other prevailing rocks of the district, namely, granite or granite-gneiss, quartzite, or the metamorphosed representative of a series of basic sedimentaries, now termed "greenstones" by the Survey.

Dr. Barlow says his investigations prove that the normal

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\* F. D. Adams, "Prel. Rep. on the Geol. of a Portion of Central Ont.," *Geol. Sur. Can.*, vol. vi. (Rept. J); "On the Igneous Origin of Certain Ore-Deposits," *Proc. Min. Assoc. Que.*, 1894. J. F. Kemp, "The Nickel-Mine at Lancaster Gap," *Trans.*, xxiv., 620.

† *Summary Rep. of the Director of the Geol. Survey of Can.*, 1901.

or type-rock associated with the deposits is a member of the gabbro family of rather exceptional character. It nearly always has traces of a broad, ophitic structure,\* and the presence of hypersthene or enstatite justifies its classification as a norite. Considerable original quartz is, at times, present. In general, the rock consists of a basic plagioclase, hypersthene, or enstatite, augite, biotite, hornblende and quartz, with smaller quantities of accessories. Associated with the nickel-bearing norite, and grading into it, is a rock which is called "micro-pegmatite." Microscopically studied, the change consists in the gradual assumption of a reddish color and an increase of quartz and feldspar. The hornblende is replaced by biotite and the plagioclase by orthoclase.

*The Nickel-Belts.*—Three main belts of these norites and associated micro-pegmatites are now recognized, designated as the Northern, Middle and Southern belts respectively. They are, at present, mapped as separate, but genetically and mineralogically they are essentially identical. The northern belt runs from the old Ross Mine (Foy township) ESE. to Bowell, where it branches. The limits of the smaller branch have not been ascertained, but the larger trends to the N. and connects with the large area of basic rocks to the W. of Lake Wahnipitae. The middle band begins in Levack township to the SW. of the northern, and runs in a southwesterly direction across Windy Lake to Trill.

The southern and most important belt begins in scattered patches in Drury township (S. of Trill), which unite and extend unbrokenly NE. for over 32 miles into Denison township, where a maximum width of over 4 miles is attained. Here it is divided into two by an intrusion of coarse "augen" granite-gneiss. The northern, or more important branch, pursues a northeasterly direction through Garson township (S. of Lake Wahnipitae). The southern branch crosses the Vermilion river and passes through Copper Cliff, where it rejoins the other.

It will thus be seen that the norite belt forms a sort of rough and somewhat disconnected ellipse, the center of which is occupied by the basin of later silicified volcanic tuffs, and surrounded by the so-called Laurentian acidic gneisses.

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\* This was found by the writer, also, to hold true as a general rule.

The unraveling of the complex, which was formerly included under the general name of "greenstone," has proved a very difficult problem. The results seem to show that it can be broken up into several series, comprising eruptives, metamorphosed sediments, and some as to whose true nature it is difficult to decide.

The relation of the granite intrusions also presents a difficult problem. These granites in many places, undoubtedly, cut some of the basic rocks in close proximity to the ore-bodies, but in other places they seem to be earlier than these.

The geologists of the Canadian Survey are now of the opinion that these granitic rocks are intruded into rocks, largely of sedimentary origin, and classed as greenstones, but that these intrusions are earlier than the ore-bearing norite. The problem is complicated by the close similarity in mineralogical and chemical composition of the so-called altered sediments and the basic eruptives, and by the excessive metamorphism which all have undergone.

The points at issue are probably not yet finally settled, and a great deal of careful work will, no doubt, be necessary before the last word is spoken. In the absence of the full Survey report, the matter is in a very unsatisfactory state, but we may confidently expect, in Dr. Barlow's final utterance, a contribution of great scientific and practical value, shedding much light on the question.

#### *Various Theories of the Origin of Nickeliferous Pyrrhotite.*

A brief statement of the views held hitherto by prominent workers in this field is appropriate here.

Dr. Bell\* brings out the following points :

1. That this area was the seat of volcanic activity, with explosive violence on a large scale.
2. The greenstones along certain lines hold an abundance of angular fragments of other rocks, especially quartzite, and this brecciated condition appears to be favorable to the accumulation of ore. (Bleazard, Stobie, Copper Cliff, etc.) In fact, the ore-masses always consist of a breccia, of sulphides and country-rock in angular and rounded fragments of all sizes.

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\* *Geol. Survey of Can.*, vol. v., 1890-91, Report F.



3. The ore-bodies are lenticular and parallel to the strike of the enclosing rocks.

4. The chalcopyrite in large masses is generally nearly pure, but the pyrrhotite is always mixed with a certain amount of stony matter. This may indicate that the former has segregated from the original mixture by some secondary process.

5. The chalcopyrite is often in the form of branching strings and partially surrounds stony inclusions. In some of the brecciated ore-deposits in Levack township, the spaces between the greenstone fragments are filled, partly with sulphides and partly with light-colored crystalline granitic vein-matter.

6. The intimate association with the greenstone appears to indicate that they have originated first from a state of fusion, but have been more or less modified by other agencies. The presence of crystals of feldspar, quartz, apatite, etc., together with laminated iron pyrites and galena, indicates the action of solutions.

7. Both clastics and eruptives have suffered extensive metamorphism.

Von Foullon\* also calls attention to the metamorphosed and brecciated condition of the diorite (more properly norite, or altered gabbro; Bell's greenstone) and to the fact that this appears to be the most favorable place for ore-deposits. Along a line of fracture which can be traced from the Dominion (Bleazard) mine to the Vermilion, very similar deposits prevail, so it must be considered that the fracture caused them. There is also a fracture-line from the Copper Cliff to the McConnell mine in Denison township, along which ore outcrops, accompanied by diorite, and a breccia of gneiss and quartz-syenite. He then adds that the ores are not deposited from solution, but are of igneous origin, because they occur in an eruptive rock.

Dr. F. D. Adams,† speaking of "The Igneous Origin of Certain Ore-Deposits," claims such an origin for both the Sudbury and Norwegian pyrrhotites in gabbro. He thinks the ores segregated according to Soret's principle, and the formation of the minerals was determined according to Fournet's series.

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\* *Jahr. der k.k. Geol. Reich.*, vol. xlii., 1892.

† *Can. Mining Rev.*, February, 1894.

Philip Argall\* also points out the faulted nature of the district. He considers that a leaching out of the nickel and copper from the greenstones in which they were originally formed, and a concentration and precipitation along favorable zones, is a more reasonable explanation than magmatic differentiation.

E. Renshaw Bush† says, the schists in Denison and Graham townships in the vicinity of the nickel-bearing greenstones carry pyrrhotite in grains and aggregates between the layers. He suggests the aqueous origin of the ores for the following reasons :

1. Because of the tendency of the sulphides to occur along planes of contact and fracture, and the impregnation of the rock near such planes.

2. Because the ore-bodies occur at the contact between structurally different rocks.

3. Because of the impregnation of the schistose areas near the greenstones.

Merritt‡ thinks that a secondary concentration is necessary to explain certain features of the ore-bodies, viz.: the presence of native copper; the "horses" of barren country-rock, cemented by ore; the "fluccan" observed across certain deposits; and the sharply brecciated nature of some of the "horses."

D. H. Browne,§ in an article on "Segregation in Ores and Mattes," seeks to draw an analogy between a pot of matte, in which he found that the nickel- and copper-sulphides tended to separate somewhat, and the Sudbury ore-bodies, where it is seen that pyrrhotite and chalcopyrite, to a certain extent, form separate masses.

Prof. J. F. Kemp|| appeals to the laws of thermo-chemistry to explain the Sudbury deposits. As is well known, in a fused mass or solution the order of formation of compounds is determined by the amount of energy (heat) which they develop

\* *Proc. Col. Sc. Soc.*, December, 1893, vol. iv., p. 407.

† *Eng. and Min. Jour.*, March 17, 1894, "The Sudbury Nickel-Region."

‡ *Trans.*, vols. xvii., 293, and xxiv., 755.

§ *School of Mines Quart.*, vol. xvi., 1895, p. 297.

|| "An Outline of the Views Held To-day on the Origin of Ores," *Min. Ind.*, vol. iv., 1895, p. 755.

on crystallizing. If, now, the gabbro magma is considered as an intrusion in which the bases have been concentrated near the walls by Soret's or some other principle, and a stream of sulphuretted hydrogen and sulphurous anhydride gases finds a way of escape along the contact, the sulphides would form in the order indicated above. This should give in order, the sulphides of iron, copper and nickel. Prof. Kemp, however, recognizes that in this case nickel should be associated with the copper, not the iron sulphide.

Professor Vogt's views on this subject are well known.\* He considers that on account of the close chemical and mineralogical relations of these deposits the world over, they can be explained only by a common general chemical process. He also emphasizes the fact that there is a regular transition from the pure ore, on the one hand, to normal rock, on the other, by a gradual decrease of sulphides. He claims further support for his theory in the facts that titaniferous magnetite is often present in the ore; that the platinum metals, which are regarded as essentially of igneous origin, are found in Sudbury; and that eruptive magmas can dissolve considerable quantities of sulphides. This last seems to be shown by the fact that basic blast-furnace slags often contain 3 to 5 per cent.  $\text{CaS}$  and  $\text{MnS}$ ; 4 to 6 per cent.  $\text{FeS}$ ; 6 to 8 per cent.  $\text{ZnS}$ , etc. His conclusion is that the sulphides are undoubtedly derived from original eruptive magmas, by a process of differentiation according to Soret's principle, influenced by other factors of which gravity may be important.

T. L. Walker† says that near the nickel-deposits the basic eruptives are more or less completely altered by metamorphism, while farther away the change is less, till practically unaltered rock is found. He also brings out the very important fact that some at least of the granites, in contact with the nickel-bearing greenstones, formerly held to be of Laurentian age, are really younger than the greenstones. This is well seen at the Murray mine, where the "younger granite" sends apophyses into the surrounding greenstones.

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\* "*Sulphidische Ausscheidungen von Nickelsulphiderzen.*" *Zeit. für prak. Geol.*, 1893, p. 125. For other papers see bibliography, below.

† "Geological and Petrographical Studies of the Sudbury Nickel District." *Q. J. G. S.*, vol. liii., 1897, p. 40.

Breccias are also formed in places by the inclusion of angular fragments of greenstone in the granite. Similar examples are seen in the granites south of the Blezard mine and the coarse-grained gneiss north of the Copper Cliff, which is essentially the same rock.

In direct opposition to the views of Professor Vogt, another eminent European geologist, Professor R. Beck\* (Freiberg) may be quoted. Professor Beck in his admirable work on ore-deposits (Fig. 18) pictures a polished section of mixed ore and rock from the Murray mine, near Sudbury, showing its brecciated nature. Such an association, he claims, could not result from a direct magmatic segregation, but must have been due to a separation of the ore, *during* or *after* the metamorphism.

Speaking of the Norwegian nickel-deposits, Professor Beck says there are weighty considerations opposed to the theory of a direct igneous origin of the ores.

1. On physical grounds it is very difficult to understand how the molten sulphides penetrated so far into the cool, surrounding rocks (schists), as Vogt figures it.

2. On the basis of a microscopical examination of the ore, he considers that the corrosion of the residue of the rock minerals in the ore is due to solution by water, especially as most of the ore occurs in very strongly metamorphosed parts of the gabbro, and that here the ore-separation appears later than the metamorphism. In Figs. 16 and 17† the relations are shown. The gabbro and norite, associated with the ore, is nearly always changed to amphibolite, or garnet-amphibolite, and the ore seems to have followed the metamorphic changes, or in some cases to have been contemporaneous with them. The conclusion is that the important concentration of the ore took place *during* the regional metamorphism, and *later* by the aqueous method.

Posepny,‡ referring to the Sudbury ore-bodies, claims that an igneous origin is a "chemical impossibility." But the contention is not borne out, when we consider that pyrrhotite is one of the commonest of the early crystallizations from the gabbro magma.

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\* *Lehre von den Erzlagerstätten*, 1st ed., 1901.

† *Op. cit.*, pages 41 and 42.

‡ "The Genesis of Ore-Deposits," *Trans.*, xxiii., 330.

S. F. Emmons\* is of the opinion that the ore has been concentrated by percolating waters along lines of faulting and brecciation.

*The Rossland Pyrrhotite-Deposits.*

By way of comparison, a word about the Rossland, B. C., pyrrhotites may not be out of place. The writer has to thank Mr. R. W. Brock, of the Geological Survey of Canada, who has been one of the chief workers in this province, for the latest data.

The ore-bodies here, while not carrying nickel in any quantity, have the same general associations as at Sudbury; and it is now fully established that they are of secondary aqueous origin.

During the trial of the suit of the Iron Mask Mining Company against the Centre Star Company, in 1899, evidence was submitted by Messrs. Clarence King, Waldemar Lindgren and R. W. Raymond as to the nature of these deposits.†

Stated briefly, the Rossland district forms part of a huge system, reaching from Cape Horn to the Arctic, and has been involved in the enormous dynamic and volcanic effects which this region has undergone from early geological times. Following the folding of the dynamic periods, the deposition of mineral matter in the fissures and along lines of weakness took place. In connection with most of the ore-bodies 3 types of rocks are represented.

1. A monzonite, often carrying pyrrhotite, and with the original structure more or less obliterated.

2. A darker, coarser-grained rock of the gabbro family, consisting of augite and triclinic feldspar, with little or no orthoclase.

3. The third is composed largely of hornblende and orthoclase. These types are doubtless local variations of the same magma, but of successive flows.

The veins are considered as distinctly of the fissure-type, and many of them are of the "shear-zone" variety—*i. e.*, consisting of a number of more or less parallel seams, with little displacement and no open fissures. The depth of the fissuring

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\* "Geol. Distrib. of the Useful Metals in U. S.," *Trans.*, xxii., 53.

† *Records of the Supreme Court of British Columbia*, 1899.

permitted a circulation of deep-seated thermal waters and a deposition of the chemicals carried in solution along the lines of weakness by a decomposition and replacement of the rock-minerals. The final result is either: (1) a deposition of the ore on one plane or fissure of the "shear-zone," or (2) a replacement of the rock between two planes, or (3) the replacement of a whole zone, and so on with endless variations. The replacement of the rock-minerals by ore (largely chalcopyrite and pyrrhotite) took place molecule by molecule, producing what is really a pseudomorph. The replacement is well seen near the Iron Horse mine, where large diallages or augites have changed to ore, while the surrounding country-rock was only partially transformed.

The replacement has, at times, been so intense that an almost solid body of sulphides (with quartz and calcite) results. In other places the original rock, more or less modified and silicified, has been only partially replaced and impregnated with ore.

Messrs. King, Lindgren and Raymond were practically unanimous in their interpretation of the phenomena of the district, which agrees very closely with that of the Canadian Geological Survey, though the more recent work shows that slight modifications of the rock-relations are necessary.

Briefly, the geology and relative ages of the rocks of the mining district are as follows:\*

1. The oldest series represented is classed as the Kootenay volcanic group, consisting of augite-porphyrates, tuffs, ash-beds, etc., of Paleozoic age.

2. Next comes a granite or grano-diorite (Nelson granite), probably Jurassic.

3. The Rossland monzonite, probably post-Jurassic.

4. Conglomerate, probably Tertiary.

5. Alkali (Rossland) granite and syenite.

The ores may occur in any of the rocks older than the alkali granite. This granite, whose main development is outside the limit of the Trail Creek sheet, is probably a tertiary eruption, and its dykes have penetrated all through the Rossland district.

As it happens, the principal mineralization is in the augite-

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\* See Trail Creek map, 1897, and *Summary Reps.*, 1896-1900, of the *Geol. Surv. Can.*

porphyrites and monzonite, probably on account of the fracturing and cutting of these rocks by dykes, together with their relative solubility. The chief factor controlling the location of the mineralization seems to have been the alkali-granite contact and its system of dykes, the eruption of which *immediately preceded the formation of the ore-bodies*. The deposits are found where the dykes are particularly abundant.

The monzonite is the chief centre of mineralization, but it is not to be considered as a volcanic neck, with the augite-porphyrates and tuffs as part of the cone. These latter are much older and are cut by the Nelson granite. The occurrence of the ore has no relationship to the contact of the monzonite, being found both *outside* and *inside* its boundaries and in the younger rocks.

While magmatic differentiation has gone on to some extent, the *sulphides are not the result of it*, as is proved definitely by the work of the Survey.

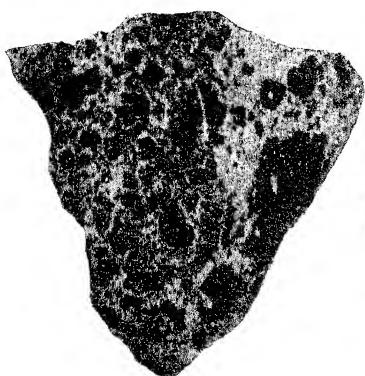
A peculiarity of the rocks in which the ore occurs is that, while sheared and fractured, they are not brecciated, the dynamic movement having doubtless taken place under an immense load.

With reference to the ore-bodies themselves, some interesting points are brought out. The ore replaces the country-rock, partially or completely, starting from some fissure or line of fissures and often fading out gradually, the only "wall" being a commercial (economic) boundary. It may end abruptly at a fissure, in some cases due to a slip, which brought an unmineralized face against the ore.

Deposits very similar to those in the eruptives are found in the conglomerate (No. 4), which cannot be regarded in any other way than as of secondary origin.

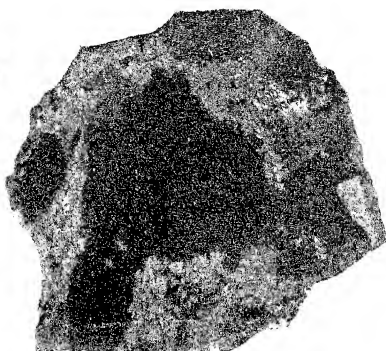
The marked similarity of the geological relations in Rossland and Sudbury help to make clear some of the more obscure points in the latter district. As the conditions of metamorphism, however, were not identical, the dynamic movements have manifested themselves in different ways, and it is not surprising that there should be striking dissimilarities. This difference is seen in the characteristic structures of the ore-bodies in the two districts. In Rossland, the fissure or shear-zone type of vein is predominant, with little or no sign of brecciation.

FIG. 1.



Mount Nickel Mine. Photograph of polished section of ore. The light is chalcopyrite and pyrrhotite; the dark is gangue and rock. Slightly enlarged.

FIG. 2.



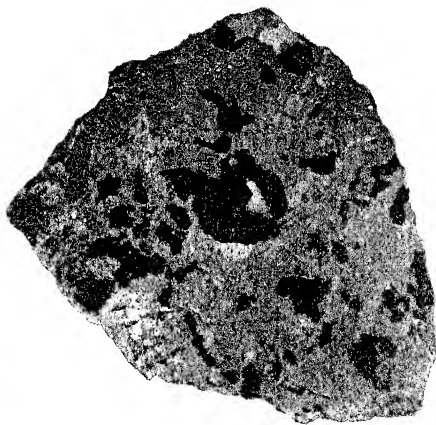
Mount Nickel Mine. Photograph of polished section of ore and rock, showing brecciated character. Slightly enlarged.

FIG. 3.



Mount Nickel Mine. Polished section of massive ore with unreplaced rock (dark). Pyrrhotite and chalcopyrite indistinguishable in the photograph. Slightly enlarged.

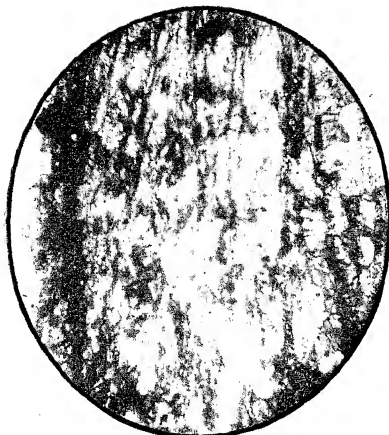
FIG. 4.



Stobie Mine. Photograph of polished section of ore (light), with angular and rounded rock-fragments (dark). Slightly enlarged.



FIG. 5.



Mount Nickel Mine. Photomicrograph of rock containing ore, showing the ore in parallel veinlets through the rock-minerals. The white is feldspar containing ore (dark), and the shaded is granular and fibrous hornblende and hypersthene. Field, 2.5 mm.

FIG. 6.



Stobie Mine. Photomicrograph of ore-rock. The dark is ore in veinlets, etc. The white is feldspar, and the shaded is hypersthene, often with ore in the cleavages. Field, 2.5 mm.

FIG. 7.



Mount Nickel Mine. Drawn from blue-print of photomicrograph, showing the ore in veinlets through the rock-minerals. The dark is mostly pyrrhotite, the white is feldspar, and the shaded is hypersthene. Field, 2.5 mm.

FIG. 8.

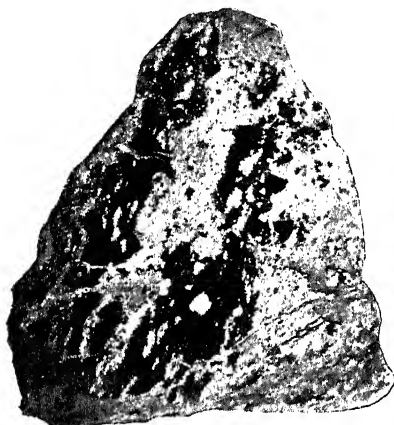


Stobie Mine. Drawn from blue-print of photomicrograph, showing the ore (dark) sharply outlined against the fresh hypersthene (shaded) and feldspar (white), and entering the cleavages. Field, 2.5 mm.

FIG. 10.



FIG. 9.



Elsie Mine. Photograph of polished section of ore (light), showing the irregular, vein-like replacement of the rock (dark). Slightly enlarged.

Copper Cliff Mine. Photograph of polished section of ore, showing the vein-like character of the sulphides. The pyrrhotite has been touched out with Chinese white to form a contrast with the chalcopyrite (dark). Slightly enlarged.

FIG. 11.



Copper Cliff, No. 2, Mine. Photograph of polished section of ore and rock, showing the brecciated character, and the vein-like nature of the ore (light), and the centers of unreplaced rock (dark). The sulphides are brought out by touching up with Chinese white. Slightly enlarged.

FIG. 12.



Elsie Mine. Photomicrograph of ore-rock, showing altered feldspar (white) with parallel veinlets of ore (dark) and irregular patches of pale-green hornblende (shaded). Field, 2.5 mm.

FIG. 13.



Copper Cliff, No. 2, Mine. Photomicrograph of ore-rock, showing ore (dark) replacing fibrous and granular hornblende (shaded), and entering cracks and cleavages. The white is feldspar and quartz. Field, 2.5 mm.

FIG. 14.



Creighton Mine. Drawn from blue-print of photomicrograph, showing the ore in veinlets, through altered hornblende, chlorite and biotite (shaded). The white is feldspar and quartz. Field, 2.5 mm.

FIG. 15.



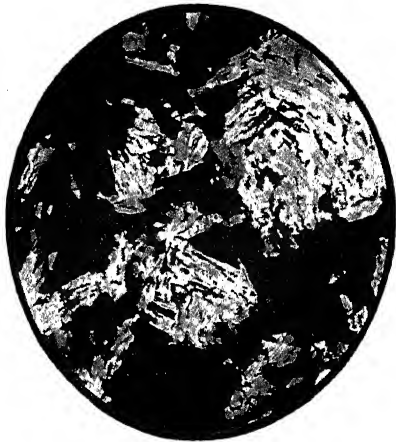
Victoria Mine. Drawn from blue-print of photomicrograph, showing sulphides (dark) between and surrounding fibers of hornblende and biotite (shaded), and entering cleavages. The white is quartz and feldspar. Calcite is also present. Field, 2.5 mm.

FIG. 16.



Gertrude Mine. Drawn from blue-print of photomicrograph, showing the sulphides (dark) parallel to the fibers of the hornblende (shaded). The white is feldspar. Field, 2.5 mm.

FIG. 17.



Gertrude Mine. Drawn from blue-print of photomicrograph, showing sulphides (dark) parallel to the cleavages of the fibrous hornblende (shaded), and following the changes of direction. Field, 2.5 mm.

FIG. 18.



Creighton Mine. Photomicrograph of ore with small residues of hornblende, feldspar and quartz, partly replaced. Field, 2.5 mm.

FIG. 19.



Gertrude Mine. Photomicrograph of ore-rock, with ore (dark) replacing the hornblende and biotite (shaded) along cracks and cleavages. The white is feldspar and quartz. Field, 2.5 mm.

FIG. 20.



Gertrude Mine. Photograph of polished section of ore, showing unreplaced rock (dark). The light is pyrrhotite and the shaded is chalcopyrite. Slightly enlarged.

FIG. 21.



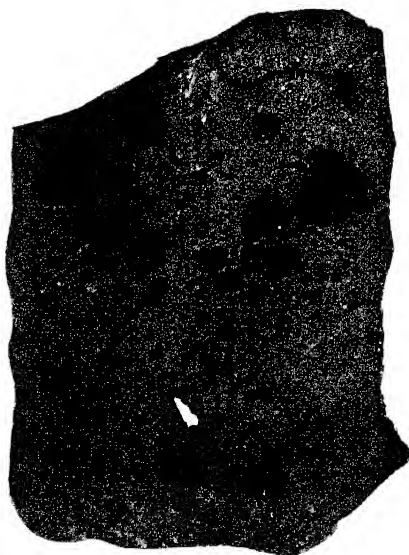
Creighton Mine. Photograph of polished section of ore, showing the vein-like nature of the sulphides. The light is pyrrhotite and the dark is chalcopyrite. Slightly enlarged.

FIG. 22.



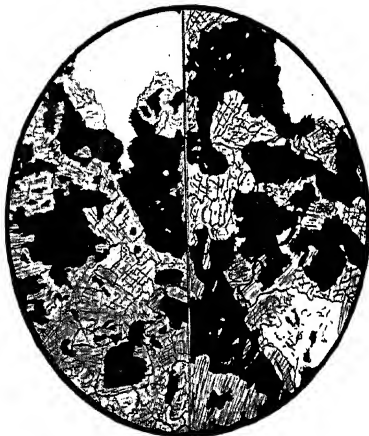
Worthington Mine. Photograph of polished section of ore, showing irregular patches of unreplaced schistose-rock. Slightly enlarged.

FIG. 23.



Worthington Mine. Photograph of polished section of ore, showing round and angular fragments of unreplaced rock. Slightly enlarged.

FIG. 24.



Copper Cliff Mine. Drawn from blue-print of photomicrographs, showing ore (dark) between the grains and fibers of the rock-minerals. Field, 2.5 mm.

FIG. 25.



Photomicrograph of ore from the East Tennessee Mine. The white mineral is calcite. The dark mineral is chalcopyrite, which has entered cracks and fissures in the calcite. Magnified 40 diameters.

FIG. 26.



Photomicrograph of ore from the East Tennessee Mine. The white mineral is actinolite. The dark mineral is chalcopyrite, which has filled even minute cracks in the shattered actinolite and has replaced it when crushed and altered. Magnified 17.3 diameters.

In Sudbury, on the other hand, where the dynamical movements probably took place under a very small load, a brecciation from faulting on a large scale and, probably with considerable movement, is characteristic. This will be brought out more clearly in speaking of the individual deposits.

This brief review indicates the wide differences of existing opinion as to the origin of pyrrhotite and the various interpretations which have been put upon the same phenomena.

### *Microscopical Evidence of the Origin of the Ores.*

The evidence on which the secondary nature of these ores is here advocated is mostly altogether different in character from that already presented. It consists in the relations between the ores and the rock-minerals, brought out by microscopical examination. As no investigation of this kind has been previously made, the results will be given in some detail. Material from all the principal mining locations has been studied, in order to avoid laying too much stress on local phenomena and to emphasize the genetic similarity of all the deposits of the district. The character of the ore-bodies in a large way will also be briefly noted as throwing much light on the general nature of the deposits.

*Sudbury District.*—The following separate localities in this district will represent its character:

1. Mount Nickel Mine, Blezard Township.—The ore consists of pyrrhotite (both coarse- and fine-grained) and a smaller amount of chalcopyrite, through which occur masses of almost barren rock, varying in size from minute particles to large "boulders," or "horses." Near what is considered as the foot-wall, the chalcopyrite increases in amount and the proportion of barren rock is greater.

The deposit is a good example of what is best described as a "*breccia ore-body*." Rounded and angular rock-fragments of all sizes, with sulphides as a cementing material, are everywhere a striking characteristic. Besides the brecciation as a whole, the rock is often broken and fractured, and little seams and veinlets of ore occur all through it in a very typical manner.

In some of the open cuts the ore is seen to run in particular streaks or bands, and may end very abruptly against barren rock.



The brecciated character, as it is seen in a large way, is also faithfully reproduced in hand-specimens. Figs. 1, 2 and 3, from photographs of polished sections, show these features very well.

While brecciation is one of the most striking characteristics of the deposit, evidences of faulting and shearing are not wanting in the slickensided and schistose nature of some of the rocks. There are a number of clayey seams, and some of the samples are decidedly fibrous. Along certain of the lines of crushing the rock is partly brecciated and partly slickensided, and here the ore occurs in veinlets and as a cement for the rock-fragments.

Under the microscope, the rock associated with the ore is black and dense, and belongs to the gabbro family. Hypersthene in fresh idiomorphic grains is very abundant, while augite and diallage are in subordinate amount. The pyroxenes have, in part, been altered to green fibrous hornblende. The feldspar forms an irregular mosaic, among the grains of which are fragments of hypersthene.\* Biotite and quartz are sparingly present.

The relations of the sulphides to the rock-minerals is well shown. Where the pyroxene is somewhat altered it becomes fibrous; and here the sulphides are best developed.

Chalcopyrite and pyrrhotite form numerous veinlets between the fibers and cleavages of the rock-minerals. In places, where not too massive, the sulphides reproduce very faithfully the fibrous structure of the hornblende, though little of this mineral may remain, so that really a pseudomorph results. In one place or another all stages of replacement can be seen, from a few dust-like particles of sulphides in the hornblende to complete replacement. Where the pyroxene is fresh, the sulphides are outlined sharply against its margin, and only enter cleavages. In this way grains of pyroxene are more or less completely isolated. Figs. 5 and 7 show veinlets of sulphides through the fibrous minerals.

The relations seem to indicate a crushed zone, in which the feldspar has yielded and formed a mosaic, and along which the

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\* The rocks from all the localities examined show a more or less pronounced migration of the dark silicates, especially secondary hornblende, into the feldspar.

solutions carrying the sulphides have acted; the greater the alteration of the rock, the more complete being its replacement.

2. Blezard (Dominion) Mine, Blezard Township.—The character of the deposit is very similar to that of the Mount Nickel, showing the same brecciation and shearing. Under the microscope, however, the rock shows a more advanced stage of alteration than that of Mount Nickel. The pyroxene has changed to greenish fibrous hornblende and chlorite, which, in some cases, makes up nearly the whole rock. The feldspar is largely labradorite, and often contains irregular fibers of hornblende. The pyrrhotite forms reticulating networks among the hornblende fibers, the veinlets of ore lying parallel to the fibers.

3. Stobie Mine, Blezard Township.—The brecciated character is here very pronounced, and the ore contains a large proportion of barren-rock masses of all sizes, as usual filled around with sulphides. Fig. 4 shows the prevailing nature of the more massive ore, with its residual angular and rounded rock-fragments.

Under the microscope, sections of rock, with a small proportion of ore, are made up largely of a fine-grained mixture of plagioclase (labradorite and anorthite) and hypersthene, with small amounts of augite, diallage and biotite; and magnetite, apatite and zircon as accessories. Quartz, in varying amount, is also characteristic. The hypersthene is quite abundant, and forms idiomorphic and rounded grains and irregular aggregates; it shows the usual microscopical characters of this mineral. The pyroxenes are, as a rule, quite fresh, but form a mosaic with the feldspar, probably due to crushing. Along certain lines, however, the pyroxenes have altered to a pale-green uraltic hornblende or chlorite.

The relations of the sulphides to the rock-minerals are striking. Following the direction of the altered zones, the sulphides are most prominently developed. A veinlet of chalcopyrite can be traced across one of the sections, following the cleavages, and between the fibers of hornblende and around the fresher grains of the other minerals (Fig. 6). In other cases the partly altered pyroxene has been more or less replaced by ore, the particles of which are elongated in the direction of the cleavages, and follow these even when they change their direction

abruptly. When found in connection with the fresher grains, the sulphides simply enclose them, following their irregularities and larger cleavages (Fig. 8).

An increasing proportion of ore is accompanied by a corresponding change in the character of the rock. The pyroxenes decrease notably in quantity and finally disappear, giving place to secondary green hornblende, chlorite and biotite. Both hornblende and biotite are fibrous, and often show deformation, due to strain. The hornblende and other secondary products, in their turn, disappear as the ore increases, and various stages of their replacement can be traced from an almost pure silicate, containing a few specks of ore, to pure sulphide. In intermediate cases the sulphides may preserve very perfectly the fibrous structure of the hornblende or biotite, with little shreds of the mineral marking its original form. In still purer ore, the dark silicates disappear almost entirely, and the feldspar is filled with inclusions of ore and changed to a confused aggregate of secondary products.

In the latter cases the amount of quartz has increased notably and remains as clear irregular patches through the ore.

4. Frood, or No. 3 Mine, McKim Township.—The ore consists of coarse- and fine-grained pyrrhotite and chalcopyrite. The latter at times occurs in almost pure masses, and is most abundant toward the walls. Barren-rock fragments occur all through the ore-body, presenting the usual breccia, cemented by sulphides.

Microscopic sections show that the rock has undergone considerable alteration. No traces of original pyroxene could be found. The secondary hornblende is bleached and ragged, and is intimately associated with the sulphides, as is the biotite, which is also plentiful. Feldspar in many cases is subordinate, while there is a notable amount of clear quartz, which is apparently secondary. Calcite also appears in small quantities. The sulphides are always found in connection with the dark silicates, and replace them to a greater or less extent.

5. Elsie Mine, Snider Township, and Murray Mine, McKim Township.—The ore-body (Elsie) is situated well within the basic rocks, the granite contact being some distance off. The ore is largely pyrrhotite, with a varying amount of chalcopyrite, which increases towards the footwall. The ore is in

places largely mixed with rock and may become subordinate in amount, even in the midst of the deposit. Nearly every piece of ore shows a characteristic breccia of rock in a matrix of sulphides, the rock being either practically barren or having veinlets of mineral irregularly through it (Fig. 9). There are many evidences of movement, as shown by the numerous clay seams impregnated with sulphides and in the schistose and slaty character of the gabbro, which is often polished and slickensided. There are also examples of what appear to be fault-planes and fissures, filled with a mixture of tremolite, quartz, calcite and ore, the latter very evidently having been introduced subsequent to the movement. The structure at the Murray mine is very similar to that at the Elsie, except that the granite contact is close to the ore-body. The granite pierces the associated "greenstones" (altered sediments?) and is undoubtedly younger.

Microscopic sections of rock with a large amount of ore show that most of the pyroxenes have changed to green fibrous hornblende, though original hypersthene more or less altered is still recognizable. The feldspar is much crushed, and is in an advanced stage of alteration, and often contains hornblende fibers, and parallel veinlets of pyrrhotite along the cleavages (Fig. 12).

The relation of the sulphides to the rock-minerals is of the usual type, *i.e.*, the replacement proceeds along and between the hornblende fibers and lines of weakness and cleavage in other minerals.

6. Mines Around Copper Cliff in Snider and McKim Townships.—This group includes the following mines of the Canadian Copper Co.: Copper Cliff, No. 1, No. 2, No. 4 and No. 5, in different stages of development. Some of the deposits are entirely surrounded by "greenstones," while others have the granite-gneiss as one of the so-called "walls." Dr. Barlow, of the Geological Survey of Canada, is of the opinion that the ore-bearing norite often occurs in more or less completely isolated patches in the "greenstones," and may consist almost entirely of ore.

No. 2 mine (and extension) affords a typical example of the "breccia ore-body." The contact is right at the granite, and the ore-bearing rock has numerous small dykes of the latter through it, and in this case, at least, it seems to be indisputa-

ble that the granite is the younger. Near the contact the granite contains a small amount of ore, and samples at times consist of both acid and basic rock impregnated with sulphides. The brecciated structure is very pronounced, and veinlets of ore occur everywhere through the rock (Fig. 11). There is also evidence of shearing, and secondary quartz-stringers are common.

The Copper Cliff mine belongs to the same brecciated type and has yielded a large number of minerals of secondary origin. (See Part I.)

No. 1 mine in places exhibits a very abrupt transition from rich ore to barren-rock. A short distance from the ore-body the rock is slightly mineralized, but on a close examination it is seen that the ore always occurs in small crevices or parting-planes, or other lines of weakness.

Under the microscope, thin sections show that the rock varies from an almost pure hornblende variety, with other minerals in subordinate amount, to more acid varieties, but in all, the dark silicates are in excess. Nearly all the pyroxene has been altered to fibrous green hornblende, though in some sections traces still remain. Quartz is often quite abundant, and the usual accessories are present. The granular nature of the feldspar, and the bent and twisted hornblende and biotite indicate severe crushing. The sulphides abut sharply against the fresher minerals, but where they are fibrous and broken, they tend to occur between the fibers and grains, more or less complete replacement resulting (Fig. 13).

As the amount of ore increases, the dark silicates disappear largely, leaving areas of clear quartz and feldspar, with the ore sharply defined against their boundaries (Fig. 24). Where the effects of crushing are most pronounced, the sulphides are best developed, and then the replacement extends from these centers along the cleavages and between the hornblende-fibers, resulting in a more or less complete substitution of ore.

7. Creighton Mine, Near the Boundary Between Snider and Creighton Townships.—This mine, which has recently been developed by the Canadian Copper Co., is perhaps one of the most remarkable nickel-deposits in the world. The ore-body is very large, and the ore is above the average in richness and purity. Mining is carried on in a huge open-cut, 200 ft. or

more across and about 100 ft. deep in solid ore, with a shaft from the bottom of the pit.

The ore, which is largely pyrrhotite with more or less chalcopyrite, however, contains many masses of barren diorite (norite) all through it, presenting a very characteristic breccia. The rock-fragments vary in size from minute particles to large "boulders," and are both rounded and angular. Where the ore is not too massive to obscure the relations, it presents much the appearance of a conglomerate cemented by sulphides, with numerous ramifying veinlets along the fracture-planes of the rock. Through the ore-body are a number of intrusions of a coarse granitic rock, which often includes fragments of the diorite (norite) breccia. This rock is itself not heavily mineralized, but where it has included fragments of the basic-rock these are more or less replaced by ore, and the replacement may extend to the acid-rock itself to a lesser extent.

From the evidence collected it would appear that this granitic rock had been intruded into, and had included fragments of the diorite at the time of, or after, its brecciation. These fragments have then been partly replaced by ore, the mineralization extending slightly to the acid-rock.

There are also two dense fine-grained diabase dykes cutting sharply through the ore-body and having well-defined contacts. These are considered as among the youngest rocks of the district. While they are usually considered to have cut through the ore after its formation, there is a possibility, if the theory of igneous origin is left out of the question, of their having existed previous to the formation of the deposits.

They do not include fragments of the ore, and their only mineralization is along fissures and joint-planes, the introduction of the sulphides being very evidently later than the intrusion. There has also been a movement in the dykes themselves since their formation, as they are composed of numerous vertical and parallel divisions with smooth polished faces, along which sulphides at times occur. If the dykes were in place previous to the formation of the ore, their compact fine-grained texture would be unfavorable to the action of the ore-bearing solutions, except along parting-planes, while the more coarsely crystalline and crushed diorite would offer an easy passage for the circulating waters.

No conclusive evidence of faulting or shearing could be found in the main pit, but in several of the test-pits sunk in the vicinity, to determine the extent of the ore-body, abundant proof was available. In these, the rock associated with the ore has been severely crushed and squeezed, so that it presents a schistose and slickensided appearance, and is often much like an amphibolite schist, with seams of ore on the parting-planes.

If these pits are in a continuation of the main body of ore, as seems probable, we have two different structures resulting from an unequal distribution of the dynamic force which caused them, and probably also due to local differences in the rocks themselves.

In the neighborhood of the Creighton mine, and towards Copper Cliff, the granite frequently cuts the basic rocks and sends dykes through them. What the relation of these rocks is to the ore-bearing variety could not be accurately determined.

Under the microscope the diorite, or altered norite, presents a somewhat gneissoid appearance, with a large part of the pyroxenes altered to fibrous green hornblende and chlorite. Quartz is present in variable amount.

The sulphides (Fig. 14) are best developed where the effects of crushing are most apparent. Some of the dark silicates have altered to indefinite aggregates of secondary products, which are spotted all through with ore and may even be entirely replaced. The veinlets of sulphides typically follow along the cleavages of hornblende and feldspar, which shows in some cases an advanced stage of alteration. Fig. 18 shows massive ore, with only small residues of unreplaced rock minerals.

The coarse granitic dykes contain abundant quartz and feldspar, and where they include fragments of the basic rock these show partial replacement, which also extends to the minerals of the acid rock.

8. Gertrude Mine, Creighton Township.—Besides the usual breccia which is very pronounced, there is abundant evidence of shearing in the different deposits under development (Fig. 20). In No. 1 pit the ore ends very abruptly against a wall of sheared diorite, which is only slightly mineralized along the shear-planes.

Along this contact and penetrating the rock breccia, a num-

ber of small granite dykes have been intruded, which are slightly mineralized. Sheared diorite, together with actinolite and other secondary minerals, impregnated with ore are noticeable in all the pits which have been opened up. As these pits are in line, it is probable that the faulting movement embraced them all, but from lack of exposures between, the fault-line could not be traced.

The brecciation and shearing, as well as the intrusion of the granite, evidently took place previous to the introduction of the ore, as all are more or less impregnated with mineral, either as veinlets or between parting-planes.

Under the microscope, the ore-rock is seen to consist very largely of actinolitic and chloritic hornblende, with other minerals in lesser amount.

The relation of the sulphides to the rock minerals is quite typical (Figs. 16, 17 and 19). Where the hornblende is of the fibrous variety, it is usually associated with ore, and all stages of progressive replacement can be seen. In samples with a large proportion of the ore, the more compact grains of hornblende, as well as the feldspar and quartz, are left as isolated patches in a ground mass of sulphides, while the fibrous variety has nearly all disappeared.

9. Victoria Mine, Denison Township.—The ore consists of pyrrhotite with chalcopyrite, which increases in quantity towards the footwall in a typical rock breccia. There is a fault passing through the deposit, and in the secondary minerals formed (actinolite, mica, etc.) the ore occurs in veinlets clearly later than the movement. There is also a considerable development of quartz and calcite intimately mixed through the ore, which cannot be considered as original igneous products.

The microscope shows the ore-rock to be the usual altered norite with the minerals of a diorite. Green fibrous and granular hornblende is predominant. One of the most noticeable features is the large amount of quartz and calcite, which remains spotted through the ore when the other minerals have largely disappeared. The replacement of the hornblende and biotite by sulphides is well shown (Fig. 15). The pyrrhotite has many hornblende fibers through it and enters the cleavages of the other minerals in a typical manner.



10. Worthington and Mitchener Mines, Drury Township.—The ore-rock at the Worthington is bounded on the S. by a band of dark schistose rock of sedimentary origin, and to the S. of this there is quartzite. The Mitchener mine, about a half a mile W. of the Worthington, is surrounded by the quartzite. Two varieties of rock are met with in the ore-bodies, namely, coarse- and fine-grained. The coarse-grained variety is sheared and slickensided, and, as a rule, contains little ore, but when it does, it is in the form of veinlets and along the parting-planes. The fine-grained rock has been brecciated, instead of sheared, and numerous rock-fragments of different sizes, surrounded and cemented by sulphides, remain in the ore-body (Figs. 22 and 23). Through the ore-body there are also numbers of small lenticular masses of schistose diorite giving the ore somewhat the appearance of an "augen" gneiss. These are evidently residual portions of the rock which has escaped replacement, and clearly show that the introduction of the ore was subsequent to the movement which caused the shearing and brecciation.

The ore at the Worthington is often very rich, and the nickel-mineral pentlandite occurs quite abundantly.

The fault which occurs at the Victoria seems to be continued through Denison township, embracing the deposits containing niccolite and gersdorffite, through the Worthington, and gradually dying out toward the Mitchener.

Under the microscope, it is seen that the original structure of the rock has been almost entirely obliterated, and green fibrous-hornblende and chlorite are the most abundant minerals, and often show the effects of severe crushing. As the quantity of ore increases, the amount of secondary products (epidote, etc.) increases also, and the feldspar is more or less altered and peppered with sulphide grains. Quartz is often present in large amount. Progressive replacement can be traced both in the irregular aggregates and fibrous masses of hornblende and biotite. In many cases, the introduction of the sulphides appears to start along cleavages, gradually spreading and uniting, till only isolated patches of rock-minerals remain, or the whole may be replaced, leaving no trace of the original structure. In this way, reticulating networks of sulphide-veinlets form all through the rock, oftentimes ramifying through the areas of

quartz and separating the individual grains. Samples of the coarse diorite containing ore present much the same features—*i. e.*, a branching network of sulphide-veinlets between the fibers and along cleavages of hornblende, etc., and separating the different rock-minerals.

11. Levack Township Deposits.—On the NW. are the large deposits in Levack township on the western side of the nickel-belt. These deposits have been thoroughly prospected and opened up to show their extent, and promise to be large contributors of ore in the future.

To the N. and W. the contact is on the large granite-area, formerly mapped as Laurentian, but which may prove to be much later. At the contact the ore-bearing diorite and granite are mixed up, and the acid-rock is impregnated with ore for a short distance, though not to the same extent as the basic-rock.

The brecciated nature of the ore-body is very pronounced, and is emphasized on the weathered exposures, where pebbles and boulders of rock stand out from the oxidized sulphides of the gossan.

The vein-like nature of the sulphides is everywhere seen, and is especially well shown in the diamond-drill cores.

As seen under the microscope, the rock consists, as usual, largely of secondary fibrous green-hornblende, and shows many evidences of severe crushing in the bent and twisted hornblende-fibers and the granular aggregates of hornblende and feldspar. The minerals, also, have a marked undulatory extinction. The feldspar is, at times, in idiomorphic individuals, giving the rock a somewhat diabasic aspect.

Good examples of the relation of ore to rock-minerals are presented in some of the sections. One shows a large fragment of fibrous hornblende which has been bent and twisted, with pyrrhotite between the fibers, following all the curves and leaving the comparatively fresh fibers intact. In other places the pyrrhotite has replaced the uralitic-hornblende to a greater or less extent, in some cases with such delicacy as to preserve the original fibrous structure perfectly. Where the feldspar and hornblende form granular aggregates, the sulphides form a mass of radiating veinlets around and between the grains, leaving the nucleus unreplaced.

12. North Range. Location, W.D. 16, Wisner Township.—On the northern extension of the nickel-belt a number of promising deposits have been opened up, but so far have not passed the “prospect” stage.

Deposits with good showing have been exploited in lot 6, concession 3, Norman township (Whistle mine); in Bowell township, W.D. 35, 150, 151 and 155, but lack of transportation facilities has been a serious drawback to more active work. The ore-rock is a fine-grained altered norite, shading to a coarser and more granitic variety to the S., and bounded on the N. by the so-called Laurentian granite.

As far as could be determined from the limited exposures, the ore-bodies are very similar to those farther S. That is, they consist of a breccia of almost barren-rock cemented by sulphides and with irregular veinlets of ore running through them. Besides the prevailing brecciation, there has been shearing, and numbers of samples of schistose-rock are found seamed with ore, especially on the parting-planes.

Microscopically, the rock is seen to be composed largely of secondary hornblende and chlorite, both fibrous and granular. Feldspar is often in idiomorphic crystals, giving the rock a decided ophitic structure. The structure of the sulphides is decidedly vein-like, some of the veinlets continuing across the sections and having numerous accessory ramifications. The ore is most closely associated with the hornblende, and follows between the fibers and grains, conforming to all the irregularities and replacing it partly or wholly, often preserving the fibrous structure very well. As the amount of ore increases, the alteration of the rock-minerals becomes more pronounced, and the sulphides spread all through the section, forming a very characteristic secondary network of reticulating veinlets.

*The Wallace Mine, Near the Mouth of the Whitefish River, Lake Huron.*—This deposit lies outside what is commonly understood as the “nickel-belt.” While essentially the same in some respects, so that it must be considered as genetically similar, it presents certain peculiar features of its own.

The mine is of historic interest, and was opened up as early as 1847 for copper. A year later it was discovered that the ore carried nickel, and a sample containing about  $\frac{2}{3}$  rock, analyzed by Dr. T. Sterry Hunt, yielded over 8 per cent. of this metal.

After a series of disasters, and the loss of trial shipments on Lake Huron, the mine was abandoned in 1867, and work has not since been resumed. The deposit occurs at the junction of two small dykes of so-called diorite, intruded in the country quartzite. The quartzite at the contact is very black and dense, and has been sheared and slickensided.

Some large masses of heavily mineralized ore-rock on the dump, which have resisted oxidation, show a decidedly brecciated structure, with sulphides (pyrrhotite, pyrite and chalcopyrite) cementing the fragments.

The junction of the dykes seems to have been the center of dynamic disturbances, which caused the brecciation and shearing, and formed a favorable place for the circulation and concentration of ore-bearing solutions on a small scale.

There seems little doubt that the ore was precipitated from solution, and the relations are very suggestive when compared with those of the Sudbury deposits.

The microscope shows that the rock has been severely crushed and presents a typical mosaic in places. The hornblende is both granular and fibrous, and gives the rock a schistose appearance. The rock is also much more acidic than the Sudbury types, and contains a good deal of orthoclase and quartz, often in the form of a micropegmatite, differing in this respect also from the others. The ore is mainly in veinlets, surrounding and penetrating the remaining rock-fragments, and replacing the dark silicates (hornblende and biotite) to a greater or less extent. As the amount of ore increases, the dark silicates largely disappear, and the feldspar is broken up into grains, enclosed by ore, and this, together with quartz, is left to a large extent unreplaced, while the remaining fragments of hornblende contain sulphides along the cleavages and between fibers.

*The Rossland, B. C., District.—The Josie Mine.\**—The micro-structure of the Rossland ore is so strikingly similar to that from Sudbury that a brief reference will be made to it.

The rock usually associated with the ore is a monzonite, more or less altered. The specimens examined were samples of vein-matter from the Josie (Le Roi No. 2) mine, and presented a

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\* Material kindly furnished by Mr. George H. Dickson, Rossland, B. C.

decidedly altered and schistose appearance, and contained pyrrhotite and chalcopyrite. Thin sections show that the rock is more or less silicified, and consists largely of secondary fibrous hornblende and biotite, and a smaller amount of feldspar with many fragments of the dark silicate through it.

The relation of the sulphides and rock-minerals is also very similar to the Sudbury examples.

They are intimately associated with the dark silicates and form veinlets between the hornblende and biotite fibers, and extend from these in the form of reticulating networks which in places wholly replace the minerals involved. Where associated with the more granular silicates and quartz, the sulphides form veinlets along the cleavages and around the grains and present the same features as the Sudbury ore, and can only be explained as due to a secondary introduction by means of circulating, ore-bearing solutions.

*The Ducktown, Tenn., Deposits.*—Figures 25 and 26\* show the relations of the ore- and rock-minerals in the Ducktown copper-deposits.

An inspection will show that these are very similar to some of the Sudbury examples. Professor Kemp considers that ore-bearing solutions entered along zones of crushing or faulting, where the material was of a more or less open texture, and replaced silicates and other minerals; the sulphides often insinuating themselves into the broken silicates and abutting sharply against the fresher specimens. In this connection he says, "It therefore seems probable that the replaced material consisted of the crushed and greatly comminuted country-rock, which in this condition would prove an easier prey to the ore-bearing solutions. Where the crushing was most severe, the large ore-bodies are found. Unless some factor of this sort exercised an influence, it seems strange that the process of replacement should cease so sharply against perfectly fresh and unchanged representatives of the presumably replaced minerals."

#### *Summary.*

The above results are given in some detail, even at the risk of repetition. But in a problem of this kind it is all import-

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\* From Professor Kemp's paper on "The Deposits of Copper-Ores at Ducktown, Tenn.," *Trans.*, xxxi., 244.

ant that the facts should be clearly presented, so as to obtain a grasp of the entire situation and avoid a narrow interpretation based on one or two occurrences, which might not be typical of the whole.

The investigations covering the whole field of the nickel-range show that all the deposits have certain prominent features in common.

1. Brecciation with accompanying faulting and shearing are everywhere characteristic. This is shown not only on a large scale, but is also corroborated by the microscopical relations.

2. The main brecciation and shearing took place previous to the formation of the ore-bodies proper.

3. The abrupt change from massive sulphides to barren-rock, so often noticed, seems irreconcilable with the theory of magmatic segregation, and it is also difficult to imagine how the included rock-fragments could retain their angular form if they were once part of, or floated in, a molten magma.

4. The ore prevailingly occurs as a cement for the brecciated rock-fragments and along shear-planes.

5. The rocks associated with the ore are all members of the gabbro family and can generally be referred to norite.

6. The rocks are more or less altered and now resemble diorite, the original pyroxene having in nearly all cases changed to a fibrous green-hornblende and chlorite. Where original pyroxene remains (*e. g.*, at the Stobie and Mount Nickel mines), the structure is markedly brecciated.

7. The effects of metamorphism and the development of secondary hornblende are most marked near the ore-bodies, and diminish away from them.\*

8. In general, the more complete the alteration of the rock, the more complete has been its replacement by sulphides.

9. The relation of the ore to the rock-minerals is practically identical throughout the whole district. In all cases the tendency of the sulphides is to occur along planes of weakness and in connection with the fibrous minerals.

Occasionally the relations of the pyrrhotite to the silicates suggests that it is an original constituent of the rock and one of the first minerals to crystallize from the magma. This is

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\* Compare Walker, *Q. J. G. S.*, vol. liii., 1897, p. 47 *et seq.*

seen at times in the less altered specimens some distance from the ore-bodies.

The amount of this apparently original pyrrhotite is, however, always very subordinate, and is probably no more than might reasonably be expected in a gabbro.

So that while a small percentage of the pyrrhotite may be original, the main contention still holds, namely, that the sulphides are essentially and predominantly secondary.

10. Secondary quartz and calcite are often present in the ore in appreciable amount, while they are insignificant or lacking at a little distance.\*

11. Sulphides are practically lacking in the rock a short distance from the ore. The rock-fragments included in the deposit are also comparatively free from ore, except in veinlets.

12. The relation of the magnetite to the rock-minerals is in marked contrast to that of the sulphides, the former being always in more or less rounded grains in the dark silicates and generally primary.

13. With regard to the relations of the sulphides to the rock-minerals themselves, several different types might be differentiated. These types are, however, all alike in kind, and several may occur in a single specimen.

(a) Replacement starting between and extending along the fibers of hornblende, often resulting in a pseudomorph of ore after this mineral.

(b) Replacement along cleavages of the more compact and less altered minerals, breaking them up into grains and, at times, forming complete pseudomorphs.

(c) Replacement between crystals and fragments of the same or different minerals, often extending into the mineral substance, when it could be classed under (a) or (b).

(d) Replacement along planes or zones of parting and shearing, the veinlets being in general parallel and ramifying through the neighboring minerals as indicated in (a), (b) and (c).

(e) In the more crushed and granular examples, where the rock-minerals are much altered, the sulphides are peppered all through them and form an intricate and ramifying network of veinlets in the rock, at times completely replacing the minerals around certain centers.

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\* Walker, *loc. cit.*

(f) The dark silicates fall the easiest prey to the ore-bearing solutions, but many instances are noted in which the feldspar suffers. This is generally the case where the crushing and shearing have been most severe and the mineral is partly or wholly changed to secondary products. As a rule, though not invariably, this change is especially noticeable where the mineralization of the rock is most pronounced.

14. A comparison of the Sudbury deposits with those of Rossland, B. C., and Ducktown, Tenn., shows many remarkable and essential points of similarity. These deposits are now proved beyond all reasonable doubt to be of secondary aqueous origin, and this, aside from direct evidence, points strongly to a similar origin for the Sudbury occurrences.

*Relation of Chalcopyrite to Pyrrhotite.*—From the massive nature of the sulphides, very little can be accurately determined as to the paragenesis of the minerals. There are, however, certain relations existing between the chalcopyrite and the pyrrhotite which may help to explain it.

1. In a number of cases where copper is predominant, the ore consists of numerous small parallel veinlets of pyrrhotite and chalcopyrite very intimately associated. This is well illustrated by examples from the Copper Cliff and Creighton mines (Figs. 10 and 21). The same relation is a prominent feature of the Rossland ore, and hand-specimens from the two localities are almost indistinguishable.

2. The chalcopyrite is usually most strongly concentrated near the outside of the deposits, especially towards the so-called footwall. As a rule, the proportion of chalcopyrite in the main ore-body is rather small, and it tends to form fairly pure masses without much pyrrhotite. Taking these facts into consideration, it seems likely that the pyrrhotite at first constituted the largest part of the ore-body. Later, due to some dynamic movement, opening up passages through the ore-bodies, especially toward the outer limits, copper-bearing solutions entered and deposited their mineral-contents among the rock-masses and partly replaced pyrrhotite along certain lines. In some deposits, as at the Copper Cliff, where copper-minerals predominate, the fracturing may have been greater and the solutions more active or concentrated.

It is also possible that the copper disseminated in the upper



part of the ore-bodies, now eroded, has been secondarily deposited by downward-moving currents, but this does not seem to have been the case to any great extent. In the first place there is very little in the way of an oxidized gossan, and the ground-water-level is comparatively near the surface. Secondly, there is little, if any, indication of this secondary change in the way of enriched sulphides which accompany the process.

Pyrrhotite is quite generally admitted to be a product of primary concentration by upward-moving solutions in a strongly reducing atmosphere, and if any considerable concentration by downward-moving waters had taken place, we should expect to find the pyrrhotite largely oxidized or removed entirely. It is, however, a notable fact that when the thin surface-covering is removed, the pyrrhotite appears perfectly fresh and with no appreciable admixture of secondary minerals, such as are formed in the process of secondary enrichment.

*Source of the Metals.*—What the exact sequence of events leading up to the formation of the ore-bodies and the immediate source of the metallic contents was, is not yet satisfactorily settled. The question involves many intricate problems, both of a local and general nature, and with the evidence at hand an authoritative discussion would be premature.

The metamorphic processes appear to have been very complicated, and are still more obscured by the later alteration due to the mineralizing solutions.

Pyrrhotite is a product of many metamorphic processes and may be formed in any of the classes designated by Lindgren—dynamo, hydro-thermal, solfataric or contact.\*

Under which head the metamorphism of the Sudbury district could be accurately classified, it is difficult to say. Indeed, it might be considered as a combination dynamo, hydro-thermal and solfataric. From a study of the history of the district, however, it is quite evident that it was a region of vulcanism and metamorphic processes on a large scale, and the immense stores of heat and energy involved could be readily available as stimulators of the circulation and chemical activity of the mineral solutions, so that the theory of an aqueous origin presents nothing at all unreasonable.

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\* W. Lindgren, "Nevada City and Grass Valley Districts," 17th Ann. Rept., U. S. G. S., 1896, Part ii., p. 90.

Hydrogen sulphide is so common a constituent of mineral and thermal springs that its presence in the mineral solutions can be assumed. From the nature of the deposits, the ore-bearing solutions were probably highly charged with alkaline carbonates and hydrogen sulphide, but with subordinate amounts of carbon dioxide.

The microscopical investigations show that the dark silicates are often bleached and robbed of iron, or else "resorbed," forming aggregates of magnetite grains. It is thus possible that a certain amount of the iron was derived from the alteration of the minerals of the replaced zone as well as from extraneous sources. The source of the nickel may, to some extent, have been the associated rocks, which contain small amounts, but it seems more probable that the larger part was derived from greater depths traversed by the circulating waters.

Many of those who are familiar with the district, and who have discarded the theory of a direct "magmatic segregation," still consider that there was a preliminary concentration of metals with the intrusion of the norite, and that the ore-bodies assumed their present position and dimensions by subsequent processes.

This may, to a certain extent, be true, and the presence of small quantities of what appears to be original pyrrhotite in the ore-rock is cited in support of this view. Moreover, this theory does not involve many of the difficulties of the first. Still, a consideration of the whole subject and the relations involved seems to indicate that this "preliminary concentration" was comparatively slight, and appeal must be made to a more distant source of the metals, probably minutely disseminated in the rocks through which the depositing solutions passed.

In conclusion, it might be safely stated that at present the whole weight of the evidence points to the secondary formation of the Sudbury ore-bodies as replacements along crushed and faulted zones, with only minor indications of open cavities.

Previous observers have naturally been impressed by the massive character of the sulphides, in which are found the minerals of the enclosing rock, and this seems to fall in readily with the idea of an igneous origin.

The clue to the interpretation of the matter, however, appears to be furnished by the leaner material, where the relations

have not been obscured or obliterated by the excessive development of the sulphides.

The universal association of these ores with essentially similar rocks is also striking. That the norite (or gabbro) has an intimate connection with the development of the ores cannot be doubted, but in just what way they are related is not clear.

After a review of the ideas held of these and similar deposits elsewhere, it will be seen that the evidence adduced in support of the igneous hypothesis can be equally well, if not better, interpreted on the basis of replacement, while many of the observed phenomena cannot be satisfactorily explained on the assumptions of the old hypothesis.

With the additional light thrown on the subject by these investigations, there seems to be no reasonable doubt left as to the true nature of the deposits, and it may be confidently expected that future work will give additional weight to the views here advanced.

Finally, with regard to my personal connection with the matter, it may be well to say that, in taking up the work, I went into the field with an open mind and for the sole purpose of interpreting the phenomena as they were presented by the facts, without being hampered by any preconceived theories. I had always been taught to regard the deposits as essentially of igneous origin, and this theory was, of course, uppermost in my mind. But as instance after instance of the relations in the field was presented in the different deposits, the "igneous" theory was not sufficient to explain the facts. Then, on calmly studying in the laboratory the material collected, the conviction became a certainty, and I present my views only for the purpose of throwing as much light as possible on these complicated problems.

Aside from the main contention that the deposits are replacements, any arguments advanced as to the actual source, manner of concentration and paragenesis of the ore may be regarded as largely tentative and as inviting discussion, by which we may hope to arrive at more definite conclusions as to the actual processes and sequence of events, culminating in these remarkable bodies of ore as we find them to-day.

*Acknowledgments.*—The writer wishes especially to acknowledge his indebtedness to Prof. Kemp, of Columbia University;

Dr. Goodwin, Director of the School of Mining, Kingston, Ontario; Prof. W. G. Miller, Provincial Geologist of Ontario; and Mr. J. Walter Wells, late Chemist to the Ontario Bureau of Mines, for assistance and kindly interest in the work.

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#### POSTSCRIPT.

On p. 16, in Table VII., there should be a line under No. 5, "tails," etc. The entries below, marked \*, refer to separate series of experiments. In Table VIII., No. 6 represents the non-magnetic portion obtained with one ampere; shows a much greater recovery than No. 5 in Table VII., due partly to a different adjustment of the separator, but mainly to the finer condition of the material after repeated feeding through the machine.

## Operations of the Hudson River Water-Power Company.

BY CHARLES E. PARSONS, CHIEF ENGINEER, GLENS FALLS, N. Y.

(Albany Meeting, February, 1903.)

ONE of the greatest factors in our industrial development is cheap and convenient power. Long-distance electrical transmission has now reached such a stage that it is feasible, and practicable, to utilize the water-power of the country which was heretofore too remote for consideration. In all of the changes of energy from its various sources to its electrical applications, water-power stands pre-eminent in point of economy. Power-users the world over have endeavored to utilize this force in preference to any other, and until long-distance transmission was found practicable it seemed that every available water-power in this section of the country was doing its full share for the benefit of mankind.

The success of each great industrial development has been due in a greater part to some one man; one who is just a step in advance of the rest in his particular field; one who is able to grasp a situation and foresee possibilities; and, above all, one who has that indomitable pluck and perseverance which sees in each obstacle one more step toward success. Such a man is Eugene L. Ashley, for it was through his efforts that the development of the greatest power on the Hudson river was made possible.

For several years prior to 1897 the various properties along the river between Glens Falls and Palmer Falls were in the hands of speculators. These men knew the value of water-rights, but they had not the ability to assemble the properties and finance such an undertaking. In 1897 Mr. Ashley began buying these water-rights, and, after two years of litigation, title was secured to all of the properties along each side of the river, and the Hudson River Water-Power Company was organized to construct a dam and transmit power to the surrounding cities and villages. A contract was let for a dam

and power-house at Spier Falls, and active operations began in June, 1900. In November of the same year the Contracting Company withdrew, and since that time all of the work has been done by the Water-Power Company.

Spier Falls was the most available site for two reasons. The materials for construction, rock and sand, were within a few hundred feet of the work, and the outcropping ledges insured the best of foundations. Mt. McGregor, on the south side of the river, furnishes a safe anchorage for the dam and secure foundations for the power-house; and the Luzerne mountains on the north side of the river give an equally good anchorage for the end of the spillway. The foundations for the entire length of the structure are built in a trench cut in the natural granite, the full width of the base. This trench varies in depth from four to twenty feet, the greater depth being due to the presence of seams in the rock. By building the dam at Spier Falls a larger reservoir was created than could have been formed at any other location, and this fact had great weight in selecting the site. The reservoir will be five miles long and about one-third of a mile wide, and one foot of water from this area will add 100,000 H.P. hours to the capacity of the plant.

### *Plans.*

The general plan (Fig. 1) shows the main dam, the coffer-dam, and the location of the plant used during construction.

The overfall, or spillway, 820 feet long, is on the north side of the river, and is separated from the river section by a wing-wall at right angles to the dam. This wing-wall is 10 feet high, 400 feet long, and is built as a protection for the tail-race.

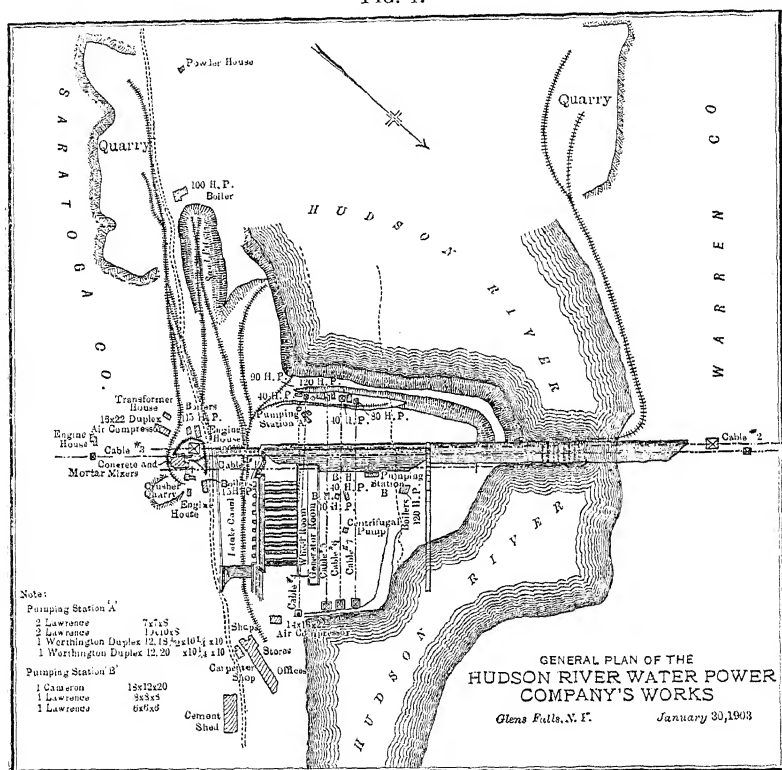
The high dam, or river section, 560 feet long, extends from the overfall to the entrance of the intake canal on the south side of the river. The canal is 400 feet long, and is formed by the intake wall on one side and the natural rock on the other. Ten feeder-pipes (Fig. 2), each 12 feet in diameter, pass through the intake wall and connect with the casings of the water-wheels. These tubes are 160 feet long.

The power-house (Figs. 2 and 3) is in the old river-bed and against the south bank, the foundations being set in a rock excavation. About 8000 cu. yds. of rock were taken out at this point and used in the masonry.



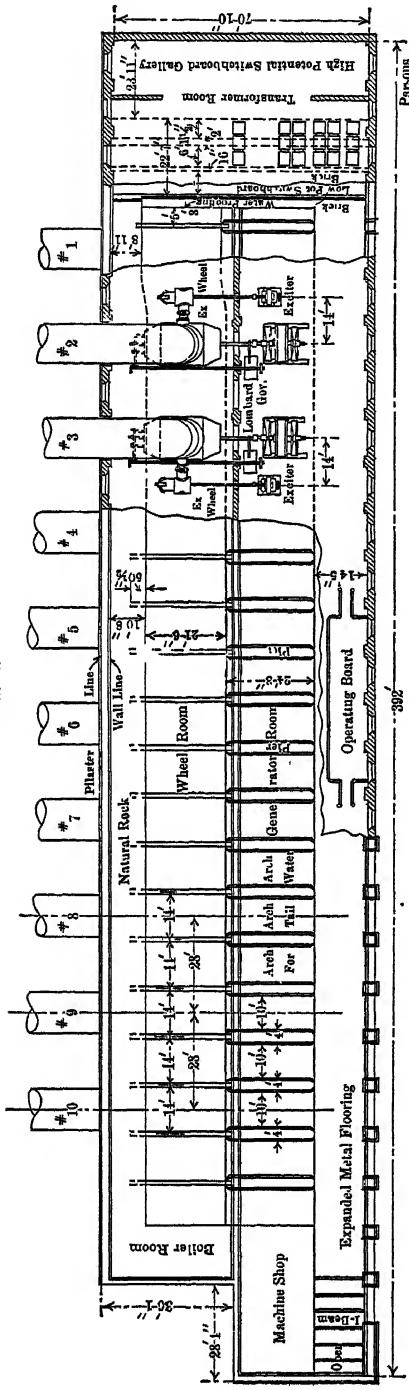
The quarry on the north side of the river was used during the construction of the overfall and part of the river section, but was abandoned in 1901. Since that time all of the stone has come from the quarry on the south side. An exceptionally good sand is found within 600 feet of the work. The concrete- and mortar-mixers, and the stone-crusher and crusher-quarry are in line with the dam and directly under the three long cableways.

FIG. 1.



Seven Lidgerwood cableways are in use; three parallel, and four at right angles with the dam. Two of these are the longest cableways in the world: one being 2140 feet between supports and the other 1660 feet. The main wire on the longest cableway is  $2\frac{1}{2}$  inches in diameter, and the others are  $2\frac{1}{4}$  and 2 inches in diameter. The third parallel cableway is 1000 feet long, and the four shorter ones are, each, 700 feet long. The head towers of the short cables stand on the upper cofferdam 65 feet above the old river-bed.

FIG. 2.



HUDSON RIVER WATER POWER COMPANY

PLAN OF

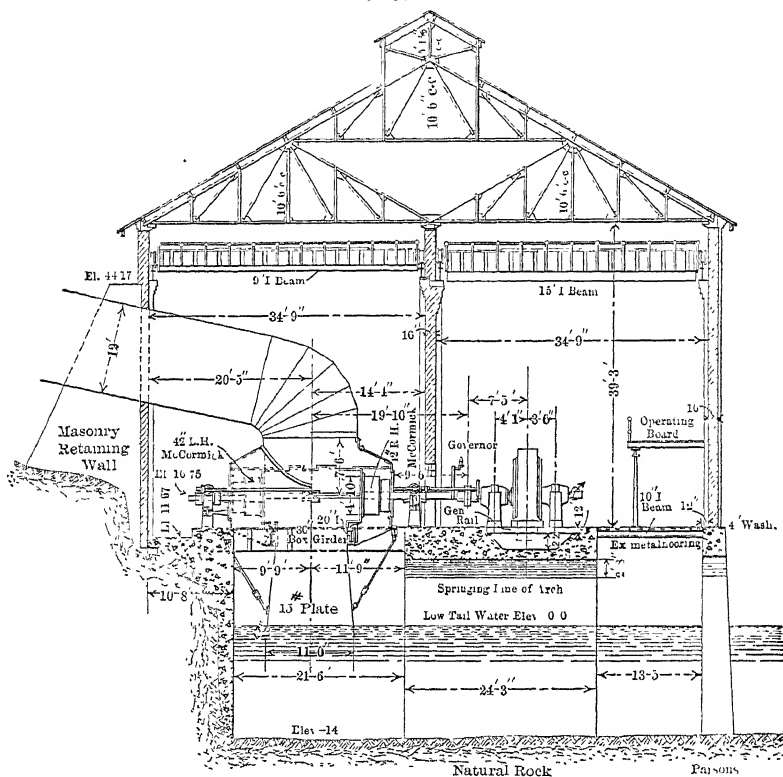
POWER HOUSE FOUNDATIONS

Glens Falls, N. Y., January 12, 1903



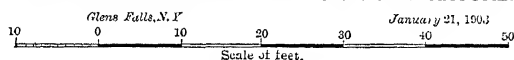
The cofferdam has been the most difficult part of the entire construction-work. It is 800 feet long and 90 feet high at a point near the south shore. It was built of round timbers spiked together, and has pockets about 10 feet square. These pockets were filled with stone and an embankment of broken

FIG. 3.



## HUDSON RIVER WATER POWER COMPANY.

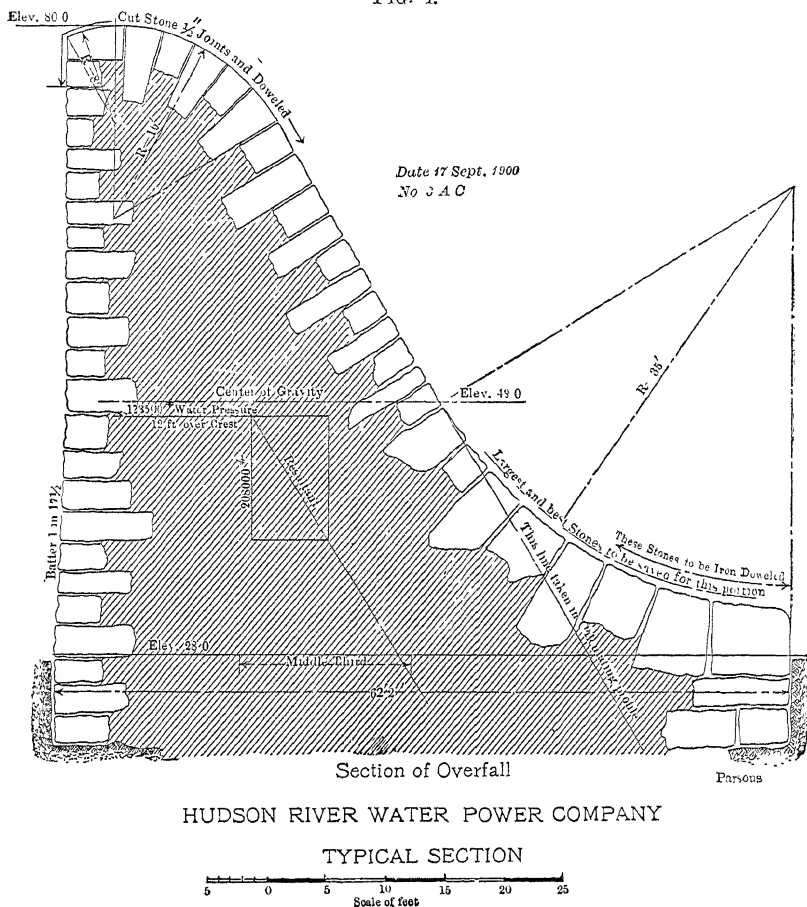
CROSS SECTION OF  
POWER HOUSE SHOWING WHEELS AND GENERATORS.



stone was made on the upper side; above, and outside of this, a heavy gravel fill made the dam complete. The main part of the cofferdam has been tried by two spring freshets. Ice-floes and logs have gone over parts of it which were then completed and it showed no signs of failure. The only trouble has been where the crib-work had not been entirely filled with stone.

The overfall (Fig. 4) is 63 feet thick at the base and 17 ft. 10 in. thick 8 feet below the crest. The downstream face is built of first-class, random range ashlar, laid with one-half inch joint. Fine pointed masonry is used over the crest and at the toe. The upper side is built of second-class ashlar laid with

FIG. 4.

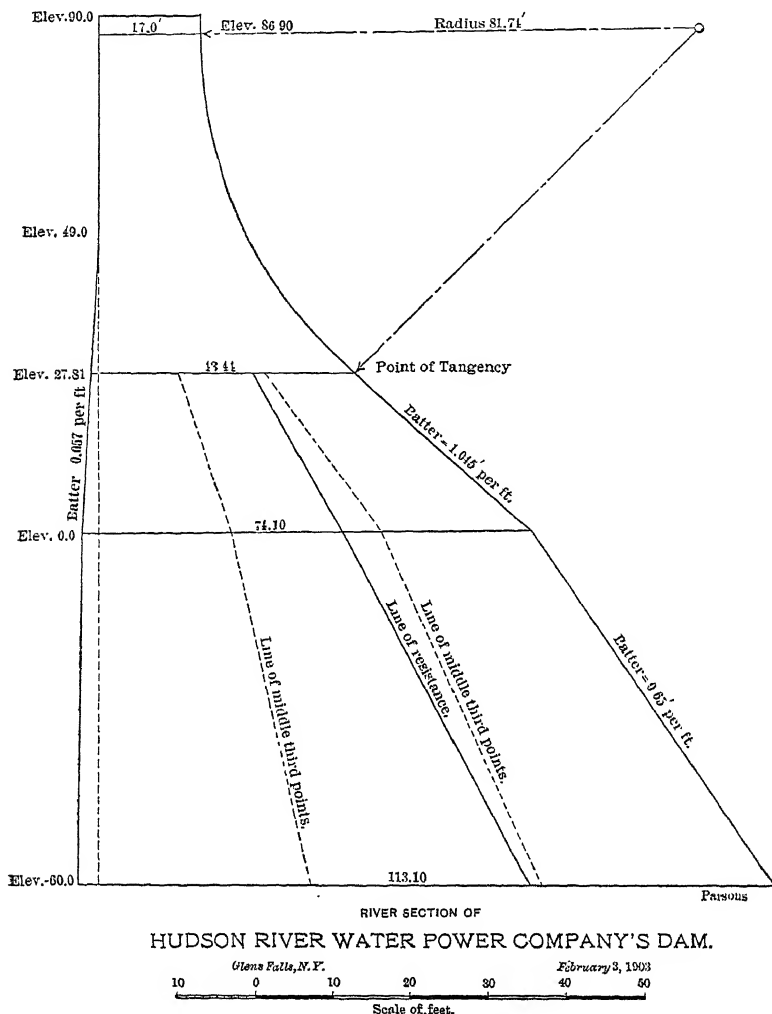


three-quarter inch joints. The filling is made of uncoursed rubble, laid in mortar consisting of  $2\frac{1}{2}$  parts sand and 1 part cement.

The high dam or river section (Fig. 5) is 154 feet high, 113 feet thick at the base and 17 feet thick at the top. Both faces are laid with hammer-dressed rubble, and the interior is filled with rubble concrete. This concrete is made of 1 part cement,

3 parts sand and 5 parts crushed stone. It is used the same as mortar, and the large blocks of rubble are set in the wet concrete. The spaces between the stones are filled with spalls and

FIG. 5.



concrete, and the whole mass settled into position with spades. The best Portland cement has been used for the entire work.

The power-house (Figs. 2 and 3) will be divided into three sections: the wheel-room, the generator-room, and the transformer- and switch-board-room. The wheel-room will be on the south side, and will extend almost the entire length of

the building. In this room are ten pairs of water-wheels, each pair capable of generating 5000 H.P. under a head of 80 feet. The wheels are set twelve feet above the water in the tail-race, and each pair has one draft-tube.

The generator-room (Fig. 3) is directly north of the wheel-room and will be separated from it by a tight partition. In the generator-room will be located ten 3350 H.P., three-phase, 2000-volt, 40 cycle, General Electric generators, each direct connected to a pair of water-wheels, and running at 240 r. p. m. There will also be in the same room, between the generators, three 265-H.P. exciters, each direct connected to its own water-wheel.

At the west end of the building will be located the transformer- and switch-board-room, 40 x 70 feet. This room will contain thirty, 1100-H.P., high potential transformers, necessary motors and blowers, and the high- and low-potential switch-boards.

On the north side of the generator-room, near the center of the building, will be located the operating switch-board. On this switch-board will be placed small controlling switches, by means of which the attendant will operate the main switches on the low- and high-potential boards, thereby placing the entire control of the plant in the hands of one man.

Directly underneath the transformer-room is the water-tight compartment for the low-potential switch-board, and directly back of this room will be the air-tight chamber, from which the compressed air will pass up through the transformers, thereby cooling them. Back of this room and on the floor above is the high-potential switch-board. This switch-board will control the electrically operated high-tension oil-switches, circuit-breakers, wattmeters, the necessary voltmeters and other measuring instruments.

#### *Excavation for the Dam and Canal.*

Mt. McGregor rises abruptly from the south side of the river, and its heavily wooded slope furnished the necessary timber for the cofferdam work. A thick layer of boulders was found in all of the earth excavation for the spillway, and their removal was so difficult that the contracting company received rock-work prices.

An idea of the difficulty of building a good cofferdam is obtained from a view of the original river-bed (Fig. 6). This

layer of boulders was about three feet thick, and underlaid by a stratum of gravel about one foot thick. The water was so swift (the fall being 4 ft. in 100) that it was impossible to clear away the boulders and gravel to reach hard-pan, so the crib-work was built on the bed, as shown, and the planking for the dam was fitted around the boulders as well as could be done in four feet of water. In spite of the difficulties, the entire leakage from the first cofferdam, 600 feet long, was handled with a six-inch centrifugal pump.

The material encountered in the excavation in the river-bed, after going down four feet, was hard-pan, and no headway could be made without blasting. All of this material was moved with derricks the first season.

The excavation for the canal and intake wall was begun in January, 1901, and 20,000 cubic yards of earth were taken out with a steam-shovel during the winter months. As the excavation progressed the highway was moved up the hill beyond the edge of the canal. A small cableway was used for conveying timber across the river for the cofferdam.

The outside wall of the power-house is built in the old river-bed, and the excavation was of the same character as that on the line of the dam, except that rock, instead of hard-pan, was found underneath the layer of boulders.

### *Cofferdams.*

The height of the masonry dam would be so great that it was not feasible to build waste-gates at the bottom, and the problem of handling the river was one not easily solved. The water had to be raised 35 feet to a new channel on the north side of the river, and any structure designed for this purpose would have to be strong enough to resist the pressure of ice and logs during flood periods. The maximum flow has been as high as 45,000 cubic feet per second, and plans were therefore made for a timber cofferdam 60 feet high.

This main cofferdam was built out 200 feet into the river, and an opening 100 feet wide was left to take care of the water the first season. At right angles to this another cofferdam 20 feet high was extended down stream 250 feet, and then back to the north shore. A bridge for handling the material from the canal excavation to the cofferdam was built across the 100-foot opening.

FIG. 6.



The River-bed.

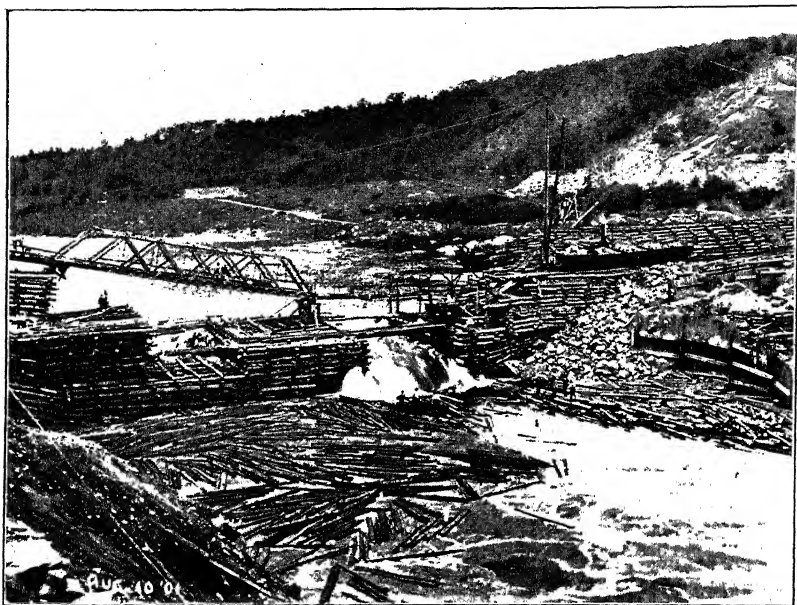
FIG. 7.



Method of Closing Last Opening in Cofferdam.

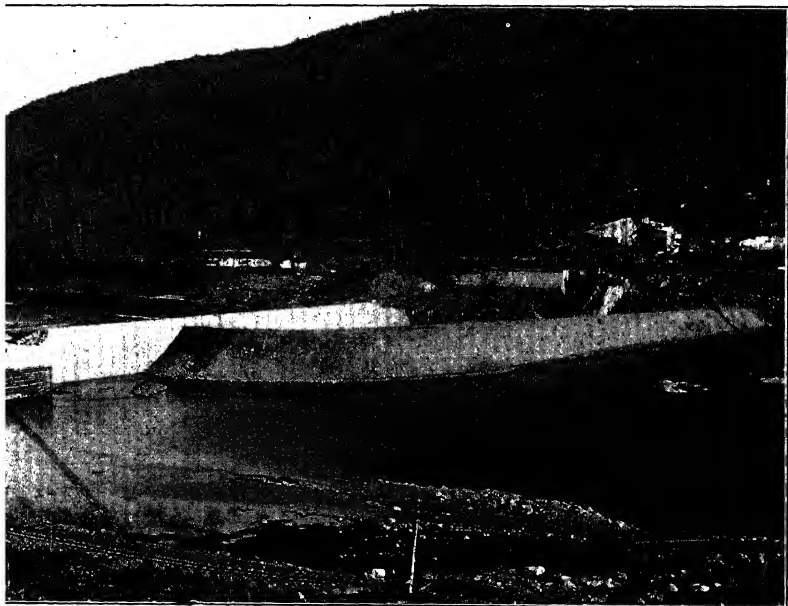


FIG. 8.



Sluiceways for Controlling River.

FIG. 9.



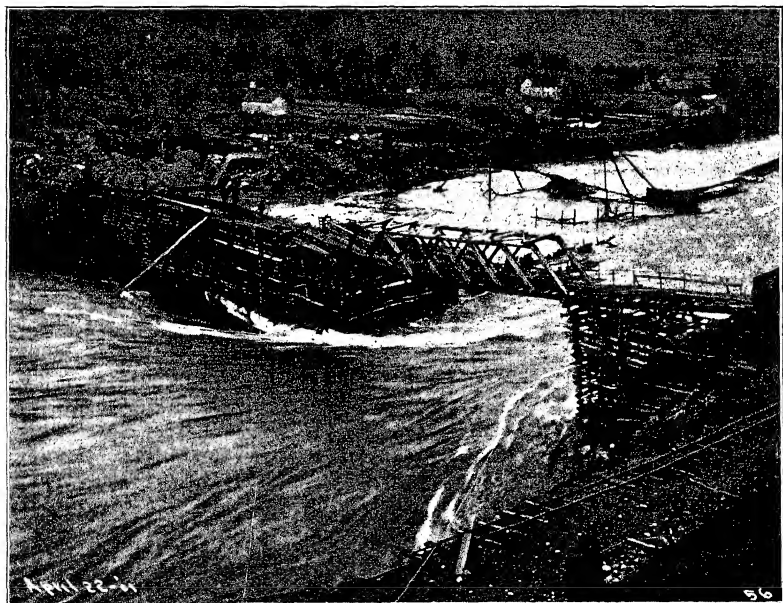
The Finished Cofferdam.

FIG. 10.



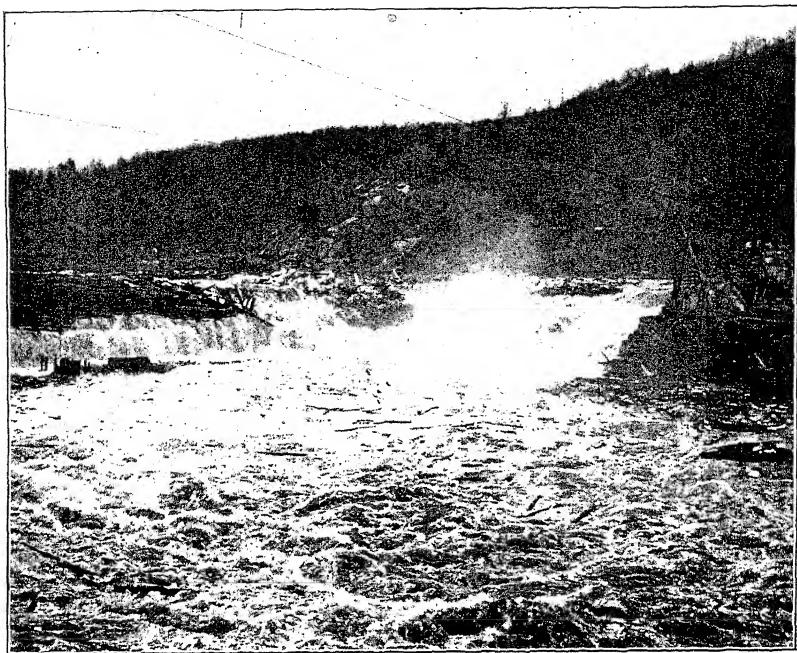
Passageway through Cofferdam.

FIG. 11.



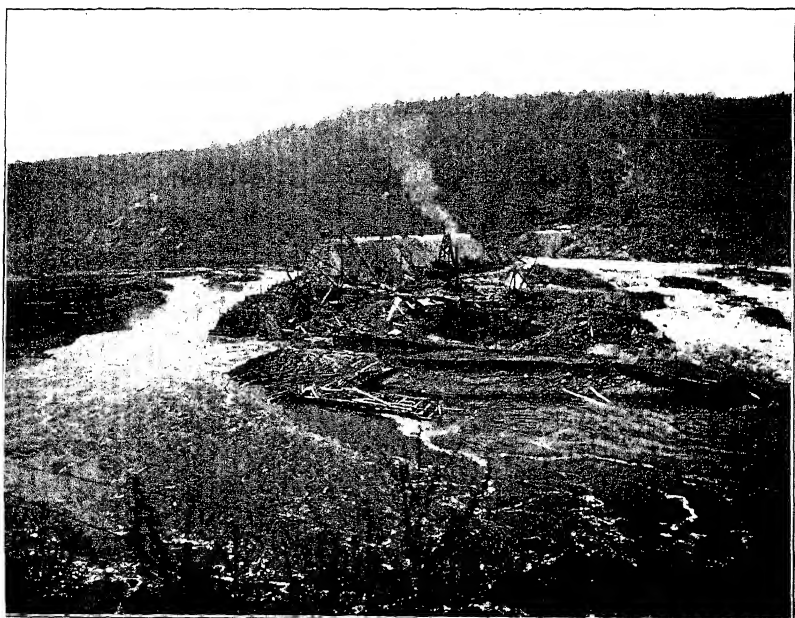
Spring Flood, 1901.

FIG. 12.



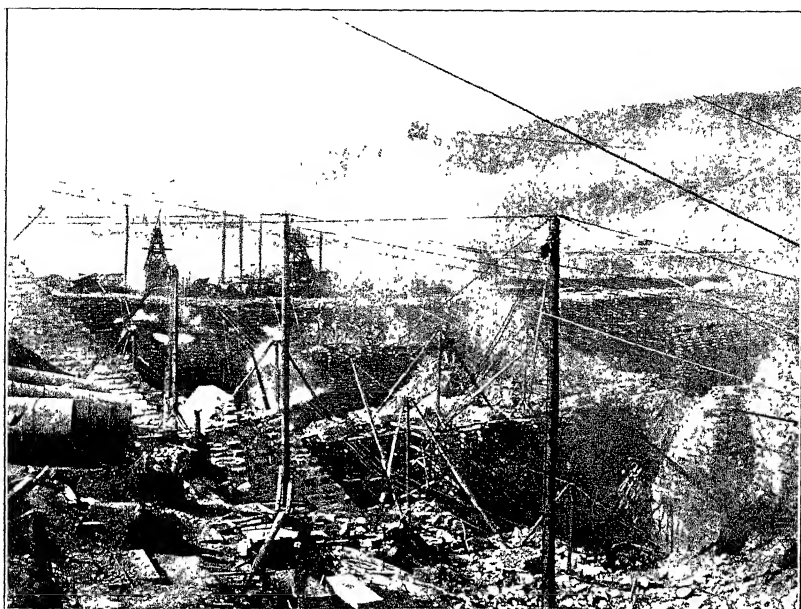
Logs and Ice going over Cofferdam.

FIG. 13.



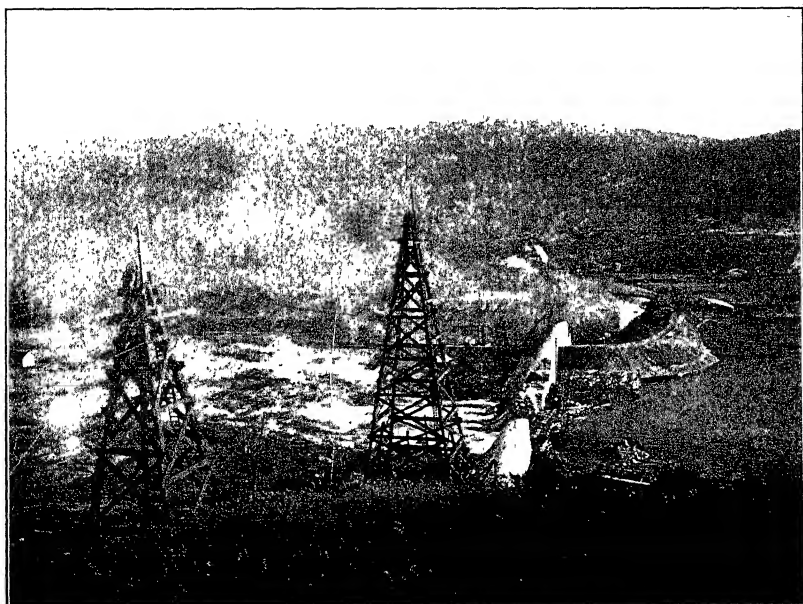
Island Formed by Freshets.

FIG. 14.



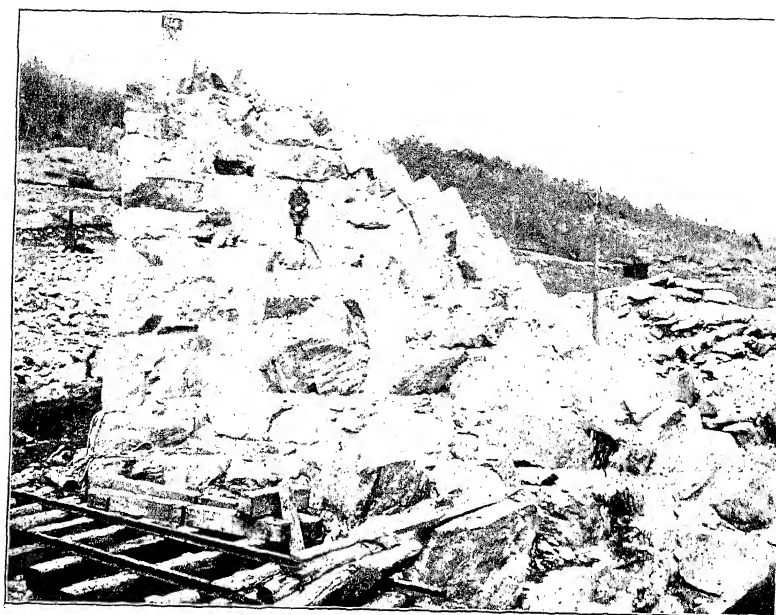
Excavation in River-bed (65 feet deep).

FIG. 15.



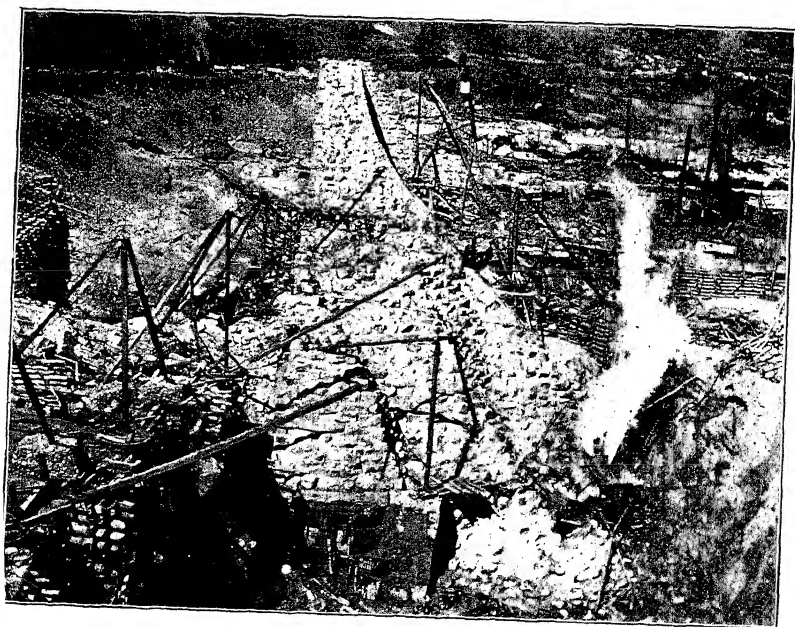
Overfall Section, 1902.

FIG. 16.



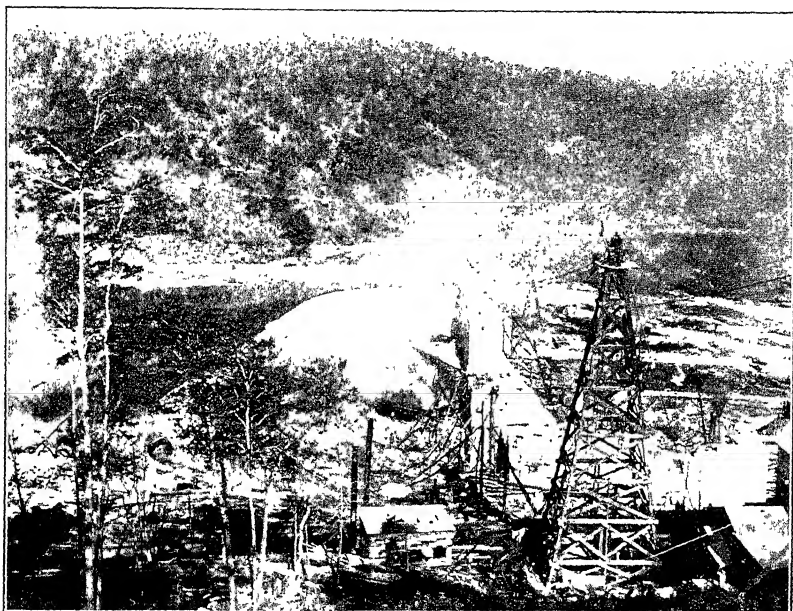
Overfall Section of Dam.

FIG. 17.



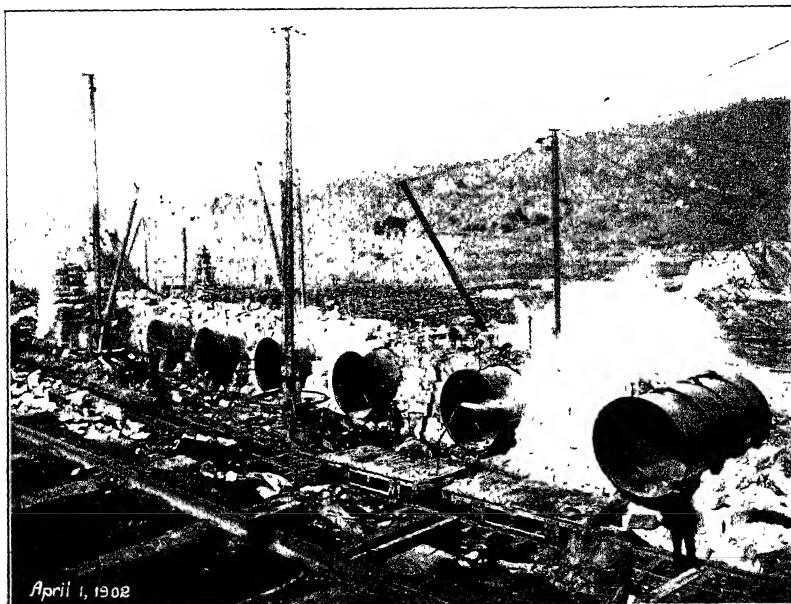
River Section of Dam.

FIG. 18.



Cofferdam and Part of Cableway Plant.

FIG. 19.



Setting First Sections of Intake-Pipes.



The spring freshets in 1901 washed out the foundation of the end (Fig. 7) of the high cofferdam, and before the dam could be carried across the river it was necessary to cut off about 60 feet of the old work. When the north abutment was taken out, the bridge was held up by six one-inch cables. The high water dug out the old river-bed for a considerable depth, and the closing of this opening was a difficult piece of work. A heavy timber crib was started on the north side, and was sunk to the bottom by using green hardwood timber. This crib was held in position by wire cables fastened to the rock on each side of the river some distance above the dam. Enough stone was then put in the crib to insure its safety, and it was built up about 6 feet above the water. The same method was used in building the crib out from the south shore. After these two cribs were built there still remained a wedge-shaped opening 20 feet wide on the up-river side and 6 feet wide on the opposite side. The crib for closing this opening was suspended above the water by means of hempen ropes, and after it had reached a height of 30 feet these ropes were cut and the wedge dropped into place.

After these foundation-cribs (Fig. 8) were in, they were filled with stone until the water rose nearly to the top. The water was raised and handled by building sluiceways at different levels. A sluiceway 20 ft. wide was built 3 feet above the top of the wedge and 25 ft. away from it, and the water was raised by filling the crib with stone until all that did not leak through the dam passed through this opening. Another sluiceway was built 6 feet above this, and the water was forced through it by closing the front of the first with six-inch plank. After these stop-planks were in, the sluiceway back of them was filled with timber and stone. The water was carried up in this manner 35 feet, where it found an outlet through the temporary archways left in the masonry near the north end of the spillway. A large sluiceway was also left near the top of the cribbing to take care of the floods.

The crib-work being finished, it was necessary to stop the leakage, and this was done by first making a heavy fill of broken stone along the upper face of the dam. This rock-fill was, perhaps, the most essential feature of the work, for it would have been almost impossible to stop some of the large leaks which appeared later, without it. When a leak started,

these stones would settle and lodge, and it was then an easy matter to hold the outer gravel-fill.

In making the embankment (Fig. 9) of broken stone, the largest ones were placed against the crib-work and the outer side of the fill was covered with the rubbish from the quarries. The gravel for the final layer was taken out in the excavation for the entrance to the canal, and it was the most suitable material which could have been found. After two months' work with the steam-shovel, locomotives and cars, the main part of the cofferdam was practically tight. The leakage to-day, under a fifty-foot head, is handled by one eight-inch centrifugal pump. The tracks have been kept on the dam for the past year, and all of the hard-pan from the river excavation has been used to strengthen the work. The structure at its greatest cross-section is 90 feet high, 250 feet wide at the base and 80 feet wide at the top.

The fall, from the bottom of the archways back to the level of the old bed of the river, is 35 feet, and it took but a few hours for the river to make a new channel after the sluiceway in the cofferdam was closed.

The smaller temporary cofferdams have been built with two rows of sheet-piling with gravel- and clay-filling. This construction has proven very successful.

During the last spring-freshet (Fig. 10) a great volume of water passed through and over the north end of the main cofferdam, and it seemed as if at last the timber-crew had more than met its match. After a few weeks, however, the water subsided until the depth through this opening was only 8 feet. The crib at each end of the 70-foot opening was rather weak, and, on account of the volume of water passing, there was no way of making repairs at these points.

A crib was constructed near the south shore and then floated out to a large rock which was directly above the opening. The main part of this crib was 70 feet long, 8 feet deep and 30 feet wide. It was built of dry chestnut timber, and, in addition to spiking, each course was braced diagonally in every direction. The up-stream timbers for a width of 8 feet projected out 10 feet from each end of the main crib, the plan being to let the main crib just fill the opening and to have these projecting ends hold it from going through.

The crib was lowered by means of four  $\frac{3}{4}$ -inch steel cables,



and no trouble was found in guiding it until just as it began to pass the upper line of the opening, when one of the cables snapped, and the additional strain broke the others. The projecting ends struck the cofferdam, and it seemed as if the crib would go through, but the diagonal bracing saved it. A large pile of broken stone was on hand for filling, and the work was begun as soon as the crib struck. During the night the river rose 3 feet, and in the morning the crib was gone. When the crib went through it took away the ends of the cofferdam, and the difficulty of closing the opening was even greater than before. The second crib was 90 feet long, and of the same width and thickness as the first. It was floated into position without a mishap, and, after loading with stone, timber was added to the top until it reached the height of the old cofferdam.

While repairs were being made on the upper cofferdam, the construction of the smaller cofferdam below the power-house was under way. After this was completed the pumps were started.

The pumping-plant consisted of five 8-inch and three 6-inch Lawrence centrifugals, two compound 8-inch discharge Worthingtons, and two 4-inch pressure-pumps. All of these pumps were needed to lower the water enough to locate the larger leaks. After these leaks were stopped only four pumps were used.

The water which leaks through the upper cofferdam is pumped back into the river above, a height of 80 feet. Eight-inch centrifugal pumps are used in two lifts. A large box was built on the side of the cofferdam 45 feet above the sump-hole, and 10 in. x 10 in. engines, direct connected to 8-inch pumps, raise the water to this level. The water is raised the remaining 35 feet by 7 in. x 7 in. engines, direct connected to 8-inch pumps. The total leakage from all of the cofferdams is about 4,000,000 gallons (534,143 cu. ft.) each twenty-four hours.

#### *Floods.*

The greatest difficulty in the construction-work has been that of taking care of the river during high water. Floods (Fig. 11) are of frequent occurrence, and at least twice a year the water has reached a dangerous point. Large outlets have been left for these floods, but, in spite of all precautions, the work has been greatly delayed by freshets. The water rose steadily until it reached a height of 30 feet in the opening be-

tween the upper cofferdam and the south shore. The scouring action of the water took out the foundation of the cofferdam running down-stream, and the entire structure fell over into the current. This let the water into the hole, but most of the machinery had been taken out and little damage was done. By directing the entire flow of the river through this 100-foot channel, the old river-bed on the line of the masonry dam was dug out by the force of the current for a considerable depth. About 4000 cubic yards of gravel and hard-pan were thus excavated free of charge, and the saving thereby was greater by far than any loss entailed by the flood.

In the early winter of 1901, a heavy rain took the snow off from the hillsides, and in a few hours the river above the cofferdam had risen several feet. There was no way of getting the derricks, engines and pumps out of the river excavation, and there was some loss of machinery. The extreme north end of the main cofferdam was in course of construction, the crib not having been filled with stone. The force of the water carried away about 200 feet of the timber work, and washed back into the hole large piles of earth and stone which had been taken out in the river excavation. When this material was taken out last summer for the second time, nearly all of the derricks and engines were recovered, and most of them were but slightly damaged.

During the winter of 1901, repairs on the cofferdam were being made, but the undertaking was too great to be accomplished before the spring flood, which came a month earlier than it has in the last twenty years. The dam withstood the action of the water, but when the great frozen mass of logs and ice came against it a small section of the uncompleted part gave way. During an entire day the logs and ice (Fig. 12) kept up an incessant battering on the main part of the structure, and great masses were carried over the top; but the cribbing stood as firm as if it had been built of masonry.

About two-thirds of the water at this time went through the old river channel (Fig. 13), but a large volume found an outlet through the gap left in the masonry. In consequence, the greater part of the work for several months was done on an island, and the only communication was by means of the cableways.

*Foundation for the Dam.*

Underlying the old river-bed was a thick layer of hard-pan; but no better site could be found, and preparations were therefore made to remove this material. The first cofferdam was completed in July, and the work of excavation was begun. The rock was uncovered about November 1st at a depth of 65 feet below the river-bed. The up-stream side of the pit became so weak that it was necessary to hold the bank with heavy timbers, fastened at top and bottom.

The Contracting Company stopped all work in November, and the pit filled with water. The sides caved, and when the water was again pumped out the hole was found to be about half filled with earth. A Lidgerwood cableway was then erected, and in January the excavation was again under way. A self-dumping bucket was used, and the material was deposited in the river channel, to be carried out by the spring freshets.

The laying of concrete was begun early in March, but the work was suspended in April on account of the floods.

The greatest difficulty has been experienced in digging out the river section (Fig. 14) and refilling it with masonry. The hole has been flooded five times, and each time a large amount of material has been deposited in the excavation. The main cofferdam has had at times a depth of 60 feet of water against it; and the leakage from this, together with the run-off in times of heavy rain, has taxed the capacity of the pumping-plant almost beyond its limit. Since July of last year the work has progressed rapidly. All of the plant which could possibly be of use was put in; and by means of seven cableways and eight derricks, working continuously night and day, the excavation has been completed and the masonry is now above the old river-bed.

*Mortar- and Concrete-Mixers.*

Large quantities of mortar and concrete are required to lay 500 cubic yards of masonry in a day, and three revolving-drum mixers are used. The concrete and mortar plant is near the three long-span cableways, and the mixed material is loaded in one-yard buckets on small flat-cars, and run under these cables. A circular track with turn-tables is used, and 500 buckets a day can be easily handled.

Stone for a No. 6 Gates crusher is quarried within 100 feet

of the plant. The tailings from the large crusher are fed through a No. 2 Climax crusher, and the material is all elevated to the bin directly over the concrete-mixers. The sand is brought in three-yard dump-cars to the sand-elevator. The small stones from the end of the rotary sand-screen are discharged into the stone-bin, and the larger stones pass to the crusher. The cement is handled in bags, and taken on a belt-conveyor to the bins above the mixers.

### *Masonry.*

The most prominent feature of a structure of this character is, of course, the masonry, but the preliminary work usually presents greater difficulties. The excavation for the overfall was begun in June, 1900, and the first stone was laid near the extreme north end on August 14th. The work progressed slowly during the rest of the season, and when the Contracting Company left it in November, less than 200 feet of the foundations were in.

The Water-Power Company bought, and began erecting, their plant during the following winter, and by the middle of May the foundation-work was well advanced. As the work progressed it was found necessary to provide means for handling material from the south side of the river, and two cableways were erected. One was used for carrying sand to the mortar-mixer, which was located about in the center of the work, and the other distributed stone and mortar along the wall. With the help of these cableways, rapid progress was made, and in August, 1901, the overfall section was half completed.

The wall now had reached such a height that it was necessary to raise the derricks on the down-stream side, and all cross-guys had to be taken off to give free passageway for the cables. Stiff-leg derricks were therefore substituted, and, although the back legs were 80 feet long, no trouble was experienced in handling five-ton loads.

This section (Fig. 15) of the dam, with the exception of the opening left for the temporary passage of the water, was completed in June, 1902. The cross-sectional view (Fig. 16) shows the character of the work. The rock near the dam is very hard to cut, there being no regular lines of cleavage. The company therefore bought a quarry at Hadley and the granite facing was transported 15 miles to the work. Through this

section of the dam, four archways were built to hold the river under control until the water could be sent through the canal. These archways, 7 by 10 feet, are 35 feet above the old bed of the river and 25 feet below the bottom of the canal. Heavy timber gates, sliding in wooden guides, will be used for closing these openings. Afterward the archways will be permanently filled with masonry.

The first work on the foundation for the river-section (Fig. 17) was done in March, 1901, when about 5000 cubic yards of concrete were placed. Operations at this point were suspended on account of floods until the following June. The work then progressed rapidly until December. The hole was again flooded, and nothing more was done until June last year.

When the main cofferdam (Figs. 9 and 18) was finished it became possible to carry on the excavation for the full width of the river. The season was well advanced, and it was necessary to use every effort to get the masonry above the flood level. Four additional cableways were erected, and work was carried on night and day, and 30,000 cubic yards of hard-pan were taken out during the month of August.

In September the earth excavation was finished, and only a small part of the rock foundation was not covered with masonry. A streak of quicksand was uncovered near the old concrete, which was laid the previous year, and the last 20 feet of the bottom was dug out four times before the masonry could be laid.

The stone, to unite the work which had been under way for two years on the north and south sides, was laid on October 9th. Since the bottom was covered seven derricks have been worked in this section, and the wall is now (Feb., 1903) 70 feet above the foundations.

In September, 1901, a small part of the intake section was completed, and during the following winter and spring the first six feeder-pipes were put in position. The first section of these pipes (Fig. 19) has four 6-inch angles, to prevent leakage along the outside. These pipes were surrounded by a layer of wet concrete, and the rest of the wall was built of rubble with cut-stone facing on the north side and concrete on the south side.

Rubble piers, with centers 8 feet apart, were built to carry the feeder-tubes from the intake to the power-house. These tubes will be completely buried in earth, to prevent ice from forming on the inside. In this climate an exposed tube would

cause trouble, as the ice would form in a thick layer around the pipe, and the first warm day would bring the mass down against the wheels with disastrous results.

The demand for power was so great last year that some immediate means of supply had to be devised. The Water-Power Company, therefore, in July acquired control of the plant of the Hudson River Power Transmission Company at Mechanicsville, N. Y., and by making improvements in the hydraulic equipment the output of the plant has been practically doubled. That there might be no delay in the construction-work at Spier Falls, due to possible coal shortage, two air-compressors, running with power from this plant, were installed. A third compressor is being built, and in a short time steam will be entirely replaced by air. Rand compressors, connected to synchronous motors with a rope-drive, are used.

When the Water-Power Company first began construction-work there were no accommodations for workmen in the vicinity, as all of the farmhouses had been abandoned after the water rights were sold. Hotels, cottages and barns were therefore built a short distance from the works. The adjoining farms, together with others owned by the company, have provided subsistence for 1500 people.

#### *Transmission Lines.*

Two transmission lines have been built—one to Glens Falls, and one to Saratoga, Schenectady, Troy and Albany. Connections have been made with the lines from the Mechanicsville plant, and by means of sectionalized circuits the possibility of any interruption in service is very remote. The power from Spier Falls will be distributed through five feeder-circuits having a carrying capacity of 50,000 H.P. The greater part of this power will be used in Schenectady, Albany, Troy and vicinity, and the longest circuit will be 42 miles.

The plant is rapidly approaching completion, and power will be delivered this coming spring. During each working month an average of 8000 cubic yards of masonry has been laid, and 9000 cubic yards of rock and earth has been moved. The completed structure will contain 180,000 cubic yards of masonry, and 275,000 cubic yards of rock and earth will have been excavated. From a comparison with other great structures of this character it will be seen that a new record in construction has been established.

## Flue-Dirt and Top-Pressure in Iron Blast-Furnaces: A Study of the Influences Controlling Them.

BY F. LOUIS GRAMMER, BALTIMORE, MD.

(Albany Meeting, February, 1903.)

THE following study of flue-dirt and the influences controlling it may be interesting to many. It is published because it represents observations at about thirty furnaces at different times. It may assist in giving an impetus to the making of similar observations elsewhere. Until ten years ago, the metallic waste in furnace-gases was not considered a problem of pressing urgency in iron metallurgy—certainly not in the blast-furnace end of it. Many plants at present ignore, or pay but little attention to, top-pressures.

Prior to the introduction of Mesabi ore, the losses in escaping gases, if considered at all, were regarded as thermal losses, or losses of fuel (except by those interested in lead, zinc, silver, etc.). In a high-priced material like ferro-manganese, such losses were, indeed, considered important. But the behavior of the Mesabi ores, coming, as they did, in practically inexhaustible quantities, could not be disregarded by the furnace-man. These ores came to stay.

The tables appended below have purposely been altered to disguise the plant, the company and the ores used. The comparative values are maintained.

In these tables, the influencing elements and conditions which are constant, or practically so, namely, size of furnace, quantity of wind, fuel used, etc., are not mentioned.

The figures represent observations consistently followed up for a long period. The weights of flue-dirt were obtained from cars on which no other dirt was thrown. The analyses and calculations are based on the best of sampling. The effects of slips are excluded. Slips should not occur at well-regulated or properly-constructed furnaces, excepting under unusual and rare conditions. Allowances are made for moisture, scrap-charged and silicon in the metal. The cutting effect of Mesabi

ore on brick-work and valve-seats, its desirable percentage and its dirtying effect on stoves, are not considered here, though closely related to the topic of the paper.

I regret being unable to include figures showing efficiency of gas-washers and briquetting appliances. Scale, rightly or wrongly, has been considered as a Mesabi ore in stating the percentages of Mesabis used.

TABLE No. 1.

Period.	Per Cent. Fine Ore.	Blast-Pressure.	Furnace.	Volume of Wind.
14 days	34.5	11.1 lbs.	A	Constant.
16 "	17.6	9.6 "	A	"

The above represents an average for each hour through the periods named. It shows an exceptionally large increase in pressure for so slight an increase in the amount of fine ore used in the charge. Some difference in blast-pressure would be expected when fine ores are used, for the interstices are smaller and the friction is therefore greater. A change of less than an inch in the diameter of the tuyeres frequently increases the blast-pressure a pound. In this case the furnace was not very well equipped for fine ores, and it was very difficult to make the same number of revolutions against augmented blast-pressure.

Data might be quoted to show that fine ores demand stronger engines and better steam economies, unless the furnace is specially designed for their use.

Table 2 shows how much more important flue-waste became upon the advent of Mesabi ores. The losses of C, D and E are small compared to many. The charges contained in the neighborhood of 25 per cent. of Mesabi ore. These figures only include the coarser particles caught in the dust-catcher, and not that which goes to the boiler- and stove-flues, up the stack, or out the bleeder- and explosion-doors. F is a furnace using much more wind. For A and B, the per cent. of iron in the dust varied between 28 and 34. For the others, between 45 and 57 per cent. A and B used no Mesabi ore or scale.



TABLE No. 2.

Ore.	Furnace.	Average Weight Downcome Dust Deposited Daily in Dust-Catcher.	Period.	Remarks.
Lump	A	4,204 lbs.	One week	A and B used about three- quarters as much wind as C, D and E.
Lump	B	3,900 "	5 days	
Partially Mesabi	C	23,960 "	30 "	
" "	D	9,632 "	30 "	
" "	E	29,344 "	30 "	
" "	F	67,200 "	30 "	

TABLE No. 3.

*Showing in Per Cent. the Relative Amount of Dust Caught in Each Compartment During a Period of Four Months.*

Month.	Sept.	Oct.	Nov.	Dec.	Average Per Cent. Dust Weighed From Each Place.
<i>Furnace V:</i>	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Dust-Catcher.....	75.7	80.2	62.3	78.2	74.1
Boiler-House Main.....	19.1	11.8	31.0	15.0	19.2
Stove Main.....	5.2	7.9	6.8	6.8	6.7
<i>Furnace X:</i>					
Dust-Catcher.....	70.3	62.5	66.2	43.9	60.7
Boiler-House Main.....	18.7	26.9	21.1	37.8	26.1
Stove Main.....	10.9	11.4	12.4	18.3	13.2
<i>Furnace Y:</i>					
Dust-Catcher.....	73.4	75.8	83.1	66.5	74.7
Boiler-House Main.....	18.4	18.3	9.4	29.0	18.8
Stove Main.....	8.1	5.9	7.3	4.5	6.5
<i>Furnace F:</i>					
Dust-Catcher.....	63.7	62.2	73.3	79.3	69.6
Boiler-House Main.....	23.9	22.1	17.1	14.2	19.3
Stove Main.....	12.4	16.5	9.1	6.5	11.1

Final average: 69.8 per cent. of the dust obtained was from the dust-catcher; 20.8 per cent. from the boiler and flues; and 9.4 per cent. from the stove-flues.

Variations in month and in furnace may be explained by one furnace having dirty boilers, or dirty stoves, or by working at a higher pressure and requiring more steam. The flues leading from the dust-catcher to the boilers and stoves being of equal diameter, the deposit of dust is a very fair indication of the rel-

ative quantity of gas consumed for heat, blowing-engines and other power. The figures show the possibilities of a large dust-catcher, kept clean, and also show that with Lake ores in modern large furnaces about twice as much gas is used for steam as for heating the blast. More refractory ores, poorer steam economies, lower furnaces, uneconomical stoves and other considerations would affect this ratio. Many European engineers estimate 60 per cent. for the former and 40 for the latter.

Another record on Lake ores shows that of the dust deposited through a long period, 67.4 per cent. was in the dust-catcher, 23.5 per cent. in the boiler-house main, and 9.13 per cent. in the stoves, which would seem to indicate that less than 30 per cent. of the gas was used in stoves here. In the cases cited, the bleeders were closed. In plants practicing slow driving and using high blast heats, the distribution of gas-dust would probably be different. It is assumed that the fine dust carried up chimneys by draught is divided in the same proportion between stoves and boilers.

TABLE No. 4.

Furnace.	Consecutive Weeks.	Tons Ore Used.	Tons Dust Weighed.	Per Cent. Loss.	Condition.
V	1st	6560	415	6.33	Ore dry, natural state.
V	2d	6634	375	5.65	Water used, ore soaked.
V	3d	5307	238	4.48	Water used, ore soaked.
X	1st	6089	351	5.76	Dry ores.
X	2d	5926	242	4.08	Wet ores.
X	3d	5791	138	2.38	Wet ores.

Furnaces X and V are alike in size and operation.

Table No. 4 shows, or seems to show, that wetting the ores is an assistance in preventing loss, and records kept over a very long period confirm this impression. These ores are wet after weighing. There is always a chance of error when using wet ores, particularly with mechanical devices, for the bucket does not fully empty itself, and the adhering ore decreases, by its own weight, the weight of the next charge.

By using the downcome dirt a second time in a thoroughly saturated condition, some have succeeded in keeping the waste through dust, even when using 50, or more, per cent. of Mesabi ore, down to 3 and 4 per cent.

The use of fine ore and downcome dirt slackens the driving and reduces the output; hence, this is not always advisable, even considering ore-economy. Before the Vaughn gun was used and when stoppages at casting-time were customary, the fine downcome dust at many places was charged in the top while the blast was off.

Wetting the ore helps to make steam and assists in keeping the stoves clean. I once filled a jute-bag with 100 lbs. of flue-dirt saturated with water, and suspended it under the bell for five minutes while the blast was on. Upon pulling it up it was not thoroughly dry. If the ores became sufficiently valuable, it might be worth while to test the advantage of putting moist flue-dirt in bags, but they would have to be charged slowly, so as not to impede rising gas-currents.

TABLE No. 5.

Furnace.	Period.	Per Cent. Mesabi Ore and Scale.	Per Cent. Metallic Contents of Burden Lost in Dust-Catcher Dirt.
A	7 wks.	38.8	2.45
B	"	40.2	2.04
C	"	37.2	1.18
D	"	27.4	.73
			Per Cent. of Total Charge Lost.
L	4 wks.	62.0	5.29
L	1 "	55.6	2.31
M	4 "	74.75	5.40
M	1 "	75.7	6.90

For A, B, C and D the last column is obtained by dividing the total metallic contents charged into the weight of iron in the collected dust.

The last column for L and M represents the loss calculated from the charge and the amount of pig-iron obtained.

TABLE No. 5a.

Furnace.	Period.	Per Cent. Mesabi Ore and Scale.	Charges.	Per Cent. Metallic Contents Lost.	Actual Yield. Per Cent.	Material Used Per Ton Metal Produced.
A	2 wks.	50	206	9.81	53.2	1.877 ton
A	"	25	218	6.68	55.9	1.789 "

Data of this kind could be furnished indefinitely. They are given to show that the metallic loss in dust increases with the

amount of Mesabi or fine ore used. The ores were dry, no dust was recharged, and the volume of wind remained constant.

The figures for A, B, C and D in Table No. 2 are small, because no account was taken of the dust carried up the chimneys of the stoves and boilers.

It was generally accepted, in 1898, as a fair statement, that 8 to 10 per cent. of the Mesabi charged in the furnace was blown out again. This was for finely divided ore; the dust being thrown away and the ore not being wet. Before special devices are adopted, it may assist us, in studying these losses, to think of the rising furnace-gases as projecting fine ore up from the stock-level in the same way that the ebullition of gas in a champagne-glass throws up a spray of wine, or a spring in the mountain shows dancing sand in the bottom of its clear depths. The furnace-gas, with pressure of from  $\frac{3}{4}$  to 4 oz., projects particles of fine ore upward; the more fine ore there is, and the greater the volume of wind, the more fines we should expect to see thrown up. Mesabi ores will pass through finer sieves than any of the ores used in large quantities before their advent. The metallic loss through dust, when using them, is greater than with the old-range ores, unless special care is taken.

It would be a valuable contribution to our knowledge if schools of technology, such as Lehigh, Case, or the Massachusetts Institute of Technology, would work up the data on this subject. They should first ascertain the fineness of different ores, and which sizes are affected by 1-oz., 2-oz., 3-oz. and 4-oz. pressures, and how far the particles are lifted. It might be of interest to fill a small model furnace with different ores, coke and stone, and note the time of passage of colored gas or water through it under a constant head or pressure. This might help to determine whether the adoption of a certain ore would increase or decrease blast-pressure.

While we know there are Mesabites and Mesabites, we have not attempted to discriminate. All those here considered are in a finely divided state.

In Table No. 6 three furnaces, using practically the same percentages of Mesabi, show great variations in flue-dirt losses. These furnaces are of the same size, and the data varying most

markedly with tons of flue-dirt collected relate to the top-pressure.

TABLE No. 6.—*Representing a Six Weeks' Run.*

Furnace.	Relative Wind-Volume.	Blast-Pressure. Lbs.	Top Gas-Pressure in Ounces.	Mesabi. Per Cent.	Tons Flue-Dirt Collected.
X	34.08	14.5	2.36	33.48	141
Y	34.66	15.77	2.63	32.25	181
Z	28.90	15.70	1.14	35.57	116

The following Tables (7, 7a, 7b and 7c) are given to show the top-pressures at different plants. They show that it is a study of prolific import, and that furnaces should be equipped with recording manometers on the top and on each main gas-branch, the one leading to stoves and the one leading to boilers. If such gauges would record suction as well as pressure, they would afford interesting data before an explosion. They might throw some light on "slips." If the gauges are properly located, they would certainly assist in showing, better than blast-gauges do, which stoves need cleaning and which are choked by a slip.

TABLE No. 7.—*Gas-Pressure, Expressed in Inches of Water.*

Date.	Furnace.	Stoves on Gas.	Relative Wind-Volume.	East Try-Hole. Inches.	West Try- or Gauge-Hole. Inches.
Oct. 18	K	2-1-4	70	5½	5½
" 19	K	"	86	7½	9½
" 20	K	"	76	9	10½
" 29	K	1-2-3	76	11	12
Dec. 3	K	1-2-3	76	14	12
Jan. 3	K	?	80	8¾	8

Top gas-pressures as high as these show throttling, inadequate gas-main areas, or choked checkers in the stoves. This particular furnace was receiving more wind than her mains were designed to carry. Such conditions result in leaky bells; and that, with the relative hot tops which furnaces have with some ores, means frequent changes of the bells. Table 7 shows either that the combustion-chamber was clinkered, or that the checkers were choked with dirt on stoves 1, 2, 3, which recorded higher gas-pressures than when gas went through 2, 1, 4. It may show that the hot-blast man did not open his valves

wide enough. In this particular instance the gas-valves were mushroom-valves enclosed, so that the gas was forced into the stoves and not brought in by suction to the extent necessary with Spearman burners. Such a recording manometer might show when the hot-blast man opened his valves too little (if that ever occurs). It might show which leg had the greater pressure, the boiler- or stove-leg. Whatever the pressure at the furnace-top is, the pressure at the entrance to the burners is influenced by the draft of the chimneys of the stoves and boilers.

It will be noted that top-pressures are influenced by wind-volume. It is suggestive that if Table 6 is correct, then furnaces making ferro-manganese and spiegel should have their top-pressures closely watched, for the loss of such expensive ores through the top is greater, in many instances, than through the slag.

TABLE No. 7a.

Date.	Furnace.	Stoves on Gas.	Relative Wind-Volume.	Gauge-Holes. Pressure in Inches of Water.			
				No. 1.	No. 2.	No. 3.	No. 4.
Oct. 18	L	1-2-4	76	4 $\frac{3}{4}$	2 $\frac{7}{8}$	4 $\frac{3}{4}$	4 $\frac{3}{4}$
" 20	L	1-3-4	88	9 $\frac{3}{4}$	9	9 $\frac{1}{2}$	9 $\frac{1}{4}$
" 22	L	1-2-3	80	7 $\frac{1}{2}$	7	8 $\frac{1}{4}$	8
" 23	L	2-3-4	96	8 $\frac{3}{4}$	8 $\frac{1}{4}$	8 $\frac{3}{4}$	8 $\frac{1}{2}$
" 26	L	1-2-3	92	8 $\frac{1}{2}$	8 $\frac{1}{2}$	8 $\frac{1}{2}$	8 $\frac{1}{2}$

Here (Table 7a) the gas-valves were of the Spearman-Kennedy type. The above furnace was shut down, the flues and stoves were cleaned, and then subjected to experiment as indicated below :

TABLE No. 7b.

Date.	Furnace.	Stoves on Gas.	Relative Wind-Volume.	Gauge-Holes. Pressure in Inches of Water.			
				No. 1.	No. 2.	No. 3.	No. 4.
March 16	L	?	104	3 $\frac{1}{4}$	2 $\frac{1}{2}$	1 $\frac{1}{2}$	2 $\frac{1}{2}$
April 2	L	?	124	5	5	5	5
April 25	L	?	124	5 $\frac{1}{4}$	5	5	5 $\frac{1}{4}$

Table 7b shows that while, under similar conditions, top-pressures may vary with the volume of wind, it is not safe to take

pressure alone as an index of the velocities of gases entering the down-take. The reader will note that we have a pressure of 5 inches with 124 revolutions as against  $9\frac{3}{4}$  inches with 88 revolutions, yet the volume of gas issuing from the furnace-top at 124 revolutions must be larger, hence its velocity of exit and dust-carrying power must be much greater.

These tables show that manometer records, if continuously kept, may be a guide as to certain conditions, but intermittently noticed are (like all unsystematic records) misleading. I attribute the difference in pressures at different gauge-holes partially to dirt in the try-hole, and partly, also, to its proximity to the downcome mouth.

It may not be inappropriate to mention here that the same furnace with increased top-pressure seems to drive more slowly. Three instances have come under my notice of a pair of connected furnaces of the same size and equipment, and similarly treated, but unsymmetrically laid out, which seemed to indicate that the furnace nearest the boiler draught-stack drove most rapidly, and that the furnace nearest the stove-stack made the best iron and its stoves gave the best heat.

Now, I do not believe, theoretically, that top-pressures influence driving; but, as boiler gas-burners are seldom manipulated (while stove-burners are often adjusted), the boiler-house gas-main will not infrequently have a lower gas-pressure recorded on the manometer than does the stove gas-main. Top-pressures, though not shown here, do influence blast-pressure. I have known the cleaning of flues to reduce blast-pressure two-pounds. Reduced blast-pressure results in less leakage, and more wind enters the furnace from a given number of revolutions than would enter it at a higher pressure. In brief, with the same number of revolutions, more coke will be burned and the driving will be more rapid.

TABLE No. 7c.

Date.	Furnaces on Lake Ores. Pressure in Ounces.						
	L.	M.	N.	O.	P.	Q.	R.
1st day.....	$\frac{3}{4}$	2 $\frac{1}{2}$	3 $\frac{3}{4}$	$\frac{1}{2}$	1	2	1 $\frac{1}{2}$
3d day.....	$\frac{1}{4}$	1 $\frac{1}{2}$	2 $\frac{1}{2}$	$\frac{3}{4}$	1 $\frac{3}{4}$	$\frac{7}{8}$	1 $\frac{1}{2}$
7th day.....	$\frac{3}{4}$	3 $\frac{3}{4}$	4 $\frac{1}{2}$	1	1 $\frac{1}{2}$	1 $\frac{1}{2}$	1 $\frac{1}{2}$
9th day.....	$\frac{1}{2}$	2 $\frac{1}{2}$	4	1	1 $\frac{1}{4}$	1	1

Roughly speaking, for those wishing to compare Table 7c with previous records, a pressure of 1 ounce is equivalent to that of 2 inches of water.

We have now established that flue-dirt loss is influenced by the volume of wind, the dryness, fineness and quantity of fine ores, and, possibly, by top-pressures. But after considering these influences, there were losses still unexplained.

In the summer of 1894 I tabulated some losses in comparison with top-dimensions, on the hypothesis that the loss varied with and proportionately to the volume of wind and the percentage of Mesabi ore used. Reducing a number of these records to a common standard, I found that the furnaces which had the smallest stock-line and downtake diameters presented losses in excess of the estimated loss as calculated. Comparing more closely, the losses seemed to vary, as they should, inversely as the downtake and stock-line area. Theoretically, they should vary as the squares of the areas, for the carrying-power of wind varies as the square of the velocity; but this is drawing the line too closely for our limited data.

However, the matter was not so simple as we hoped. A construction in accordance with these deductions was disappointing. There are other considerations, and the knowledge must be differently applied. With a larger downtake mouth there is a greater likelihood that ore will slide from the bell into the dust-catcher, unless the stock-line is kept lower, or the downtake raised. This, and what I am about to say, emphasizes the need of a positive signal (such as an electric bell or a shining bulb) to show when the furnace is full. In short, an electric stock-gauge should be used, particularly with automatic tops and fine ores.

Of course, making the stock-line diameter greater would help, but this would in many cases conflict with conditions deemed necessary, as expressed in an article in the *Iron Trade Review*, Feb. 20, 1902. A poorly tapered stack causes numerous slips with Mesabi ores, and one slip may cost many dollars, as it may throw out from 10 to 120 tons of ore.

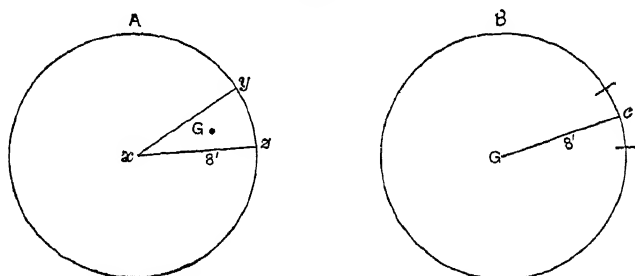
The idea of an annular ring seemed to be a good one, as it does away with the objection of one large opening, and conduces to a product of regular quality and to low fuel-consumption. This ring should be situated high up. Indeed, the mouth of the



bottom downcome should be at least 6 (and better 8) feet above the level of the drop-position of the bell. Many openings, though of greater area and producing a lower exit-velocity of gas, may cause greater loss of dust, if situated lower. For example, take two furnaces, A and B, each with a stock-line of 16 feet; A, with an annular opening, has a downcome area of 50, and B of 25 (these are entirely relative units); then the velocity of gases from A would be expressed by  $2 V$ , while the velocity from B would be  $4 V$ . Compare two circles, A and B. A has continuous downcome area. It is an annular opening. B has an opening at  $c$ . The velocity in A =  $2 V$ , and the velocity in B =  $4 V$ .

Assuming the sector  $x y z$ , in A, to be a triangle, the center of gravity of the area influenced by the suction of the downtake is at G,  $\frac{1}{3}$  of the radius, or 2.66 feet, from the arc  $y z$ .

FIG. 1.



In B, the average particle of dust travels the distance of the center of gravity (G) from the mouth ( $c$ ), or 8 feet, which is the radius of the stock-line.

If the carrying-power varies as the square of the velocity, then the relative loss of dust in two furnaces so constructed would be for A,  $\frac{2.66}{4} = .666$ ; and for B,  $\frac{8}{16} = .50$ . Or A would throw over  $\frac{.666}{.500} = 1.33$  times as much.

So, unless numerous or continuous openings are situated high up, numerous or continuous openings of even larger combined area will frequently cause greater loss, because a larger dust-area is influenced by the mouth's suction. In trying to avoid flue-dirt loss, some tops designed in the past five years

have interfered with good distribution, and that is of immense importance, even with Lake ores. Of late, this fact has been too often forgotten.

In a downtake, going out (as it should) at the top in the center where a double bell is used, the law of inverse areas and loss of dirt would hold.

The fineness of each ore should also be determined, and its elevation of projection under pressures from 1 ounce to 4 ounces ascertained. This law would hold in an annular ring if the opening were high enough, and this is the preferable location of exit with automatic fillers.

A few mistakes are quite frequently made in automatic tops. The stock-line is sometimes carried half-way up the downcome opening. In such cases there results either a rolling of ore down the downtake by gravity, or a throttling of the exit area in those furnaces where the downtake is horizontal for some distance before dropping. This loss very frequently occurs in starting up with inexperienced top-fillers who are afraid to go near the furnace, which is slipping, as most of the modern, high, automatically filled furnaces do.

The tables in this paper are but samples of data that might be obtained. The flow of furnace-gases is a problem requiring formulæ somewhat resembling those which men of nimble minds like to evolve in studying the ventilation of mines. The issuing of gases from the furnace-top presents a problem with some resemblance to the problem in the jig, where the settling of material in water is influenced by size of lump, specific gravity, liquid depth, etc.

The reciprocal corollary to studies of outflowing gases in the downtake is the study of the inflowing air at the tuyeres.

Some light on the multiple-tuyere controversy may be gained by modifying our diagrams, but the subject is not properly treated in this monograph.

I note in Mr. T. W. Robinson's\* contribution of 1888 that he advances the view that the tension of gases assists reduction.

With the ores in use at that time, which were relatively lumpy,—for all old-range ores may be so regarded when compared with Mesabis—high top-pressures would not have been

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\* "The Effect of Velocity and Tension of Gases on the Reduction of Ores in the Blast-Furnace," *Trans.*, xvii., p. 282.

accompanied by so much ore-loss, and it is likely that, as a reducing influence, it was of more moment than with finely divided Mesabis.

The pressure Mr. Robinson used in one set of experiments was 2 lbs. to the square inch, corresponding to what we should expect to find in an 85-ft. furnace, 11 ft. down from the stock-line. We can readily believe that, as all solids have pores, pressure assists gases in entering these pores and thus contributes to more intimate contact.

I tested this idea with some Cornwall ore. A Mr. Blenkinsop, of Wales, patented a process for roasting sulphides. His idea was to cool roasted ore under high pressure. Our results showed that where we could reduce sulphur to 0.8 per cent. in the Gjer kiln, when cooled under 8 lbs. pressure it was reduced to 0.2 per cent.

But our results varied with the size and the nature of the lump, and also with blast-pressure. This inclines me to the opinion that while augmented top-pressures may assist reduction when using lump or old-range ores, their use is not necessary with finely divided ores like Mesabis. In our endeavor to make a tapered stack we may obtain high top-pressures; but they should be an incident, not an aim. With poor machinery, leaky valves and a poor master mechanic, high top-pressures with Mesabi ores are a very distinct disadvantage. They should usually be avoided with fine ores, at all but the best planned and best built plants.

While it is possible that the low fuel-consumption, in the two furnaces quoted with the highest top-pressure, may have been due to gas-tension, I am more inclined to believe that the high top-pressure, in augmenting the blast pressure, prevented the same amount of wind being given to the furnace. Consequently the wind it did receive was at a higher temperature.

With perfect intimacy of contact, the rapidity of solution might be said to be represented for a homogeneous material by the ratio

$$\frac{0.5236 D^3}{\pi D^2} \text{ or } \frac{0.5236 D}{\pi} = T,$$

“D” being the average diameter of the particle acted on by the gas or liquid. This fraction means that as the surface exposed varies as the square of diameter and the weight as the

cube, the larger a particle the more time it will take to dissolve a given unit of weight.

The more lumpy the ore, the more valuable gas-tension becomes, and more time is necessary for reduction. And the converse holds: The finer the ore and more perfect the intimacy, the shorter the time of exposure that is necessary for reduction. It would therefore seem that Mesabi ores require less and not more time of exposure for reduction.

The use of furnaces 100 feet high for fine ores would seem to indicate that they required more time or, at least, as much time for reduction. That high furnaces are necessary to accomplish this is questionable. In an experiment a tin-cup, filled with marbles of sizes 9, 13, 19, contained, in units of marble-weight, respectively, 10, 8.75 and 7.75.

An arithmetical computation shows that a cylindrical measure—56 units in diameter and 92 units deep—will contain :

Size of Marbles.	Area Surface Exposed.	Weight of Contents.
13	8.2	21.5
23	4.2	19.5

These figures are entirely comparative. I appreciate that figures given in hand-books do not bear out these relations, for they give the weight of a bushel of coal for larger sizes as being greater than for smaller sizes. Nor do the figures obtained from Lake superintendents and managers regarding space occupied by Old Range and Mesabi ores in the load confirm our experiments.

But it is not to the size of the vessel in which the mixture is placed that we look for large outputs, but in intimacy of contact. It is along these lines that strides have been made.

Fine ores demand more wind, as they are (let us say) more readily dissolved. We can therefore push the plant more rapidly. Large amounts of wind, to be properly used, demand fine stock; but fine stock requires more pressure, and that means that the engines and boilers must be economical and powerful.

These well-known facts have been ignored of late in the pursuit of labor-saving devices and because of other and more important matters, such as facilities for obtaining coke, etc. I have purposely ignored chemical and thermal aspects.

## Water-Hoisting in the Pennsylvania Anthracite Region.

BY R. V. NORRIS, WILKES-BARRE, PA.

(Albany Meeting, February, 1903.)

THE removal of mine-water by hoisting in tanks instead of pumping, while somewhat a reversion to the methods of the ancients, has come very rapidly into favor in the anthracite region of Pennsylvania during the past few years. In fact, so much so, that at the present time there are at least eight large collieries at which all the water is hoisted, and six more plants are in preparation.

The particulars of the plants in operation and under construction, as far as decided on, are given in the tables on pages 107 and 108.

Besides these, the Delaware, Lackawanna and Western R.R. Co. is constructing a hoist of large capacity for a depth approximating 500 feet, to be located in the Keyser Valley, near Scranton. The hoists will be driven by electric motors, geared directly to the sheaves. The Lehigh Valley Coal Company is planning a large hoist for the Packer collieries in the Shenandoah district, but no work has been done on this as yet.

The earliest regular hoisting, I believe, was done by means of semi-cylindrical tanks (Fig. 1) at the Nanticoke collieries of the Susquehanna Coal Company. These tanks were attached under the regular shaft-carriages, taking in water through six large clack-valves in the bottom, and discharging through an end-gate opened by a lever which was operated by a guide-piece on the shaft head-frame. These tanks were designed July, 1880, by the late J. H. Bowden, one of the very early members of the Institute. Similar tanks are still used in emergencies at the Nos. 1, 2, 3 and 6 shafts of the company. The objections to their use were: that water could only be hoisted during the night-shift, or when the shafts were not in use for hoisting coal, thus requiring a very large sump, and greatly limiting the water capacity of the plants; that the alternate wetting and drying:

*Water-Hoists in Use and Under Construction in the Anthracite Region of Pennsylvania.*

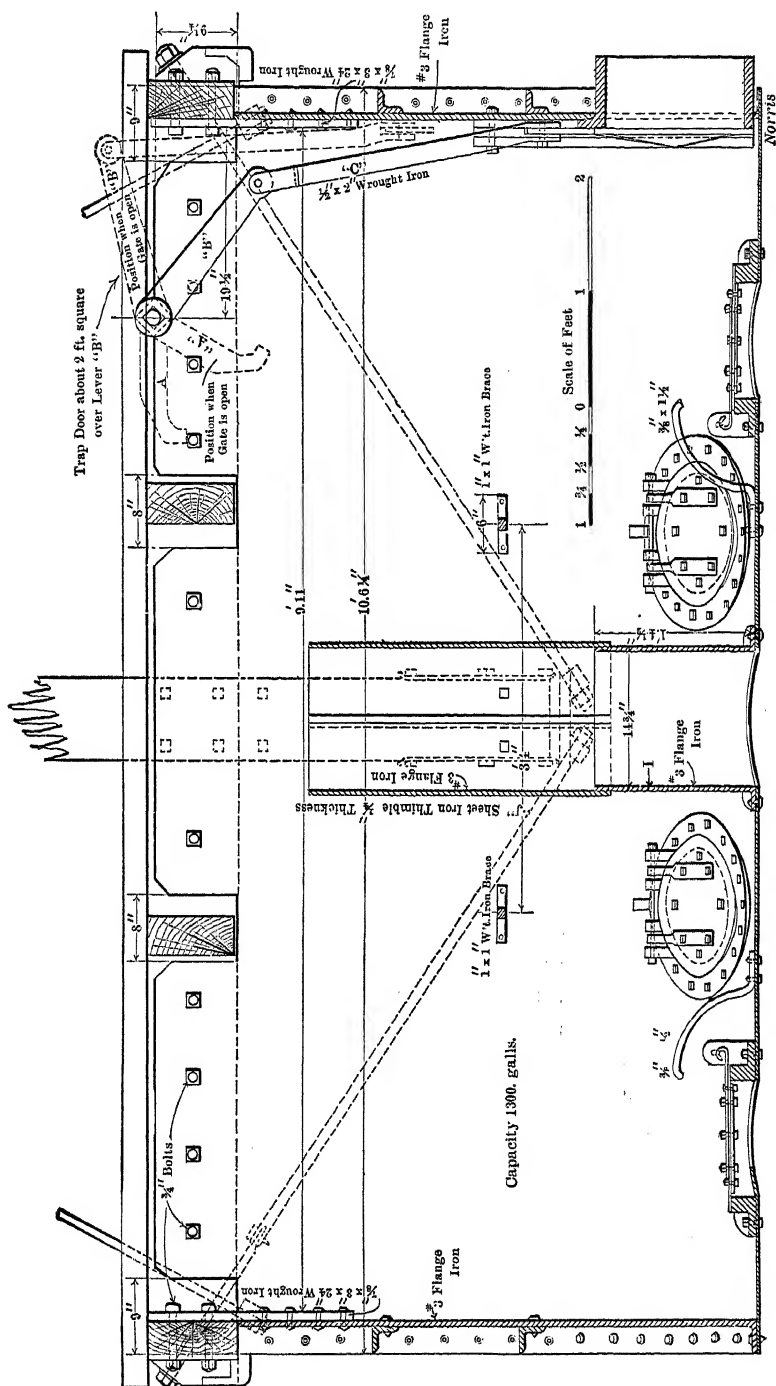
Water-Hoists in Use January 1, 1903.												
Location.	Lytle Coal Co. Shaft.			William Penn No. 2 Shaft.		Philadelphia & Reading Coal & Iron Co.					Lincoln Colliery.	
	Water-Hoist.	Coal Emergenc-y Tanks.	Coal Emergenc-y Tanks.	Water-Hoist.	Coal Emergenc-y Tanks.	Gilberton Shaft.		Maple Hill Water-Hoist.	Silver Creek Water-Hoist.	New Shaft Water-Hoist.	Old Shaft Water-Hoist.	Water-Hoist.
Holst.....	960 ft.	1500 ft.	1500 ft.	953 ft.	953 ft.	1070 ft.	1070 ft.	676 ft.	904 ft.	739 ft.	780 ft.	900 ft.
Size of engines (pair).....	32" x 48"	36" x 60"	36" x 60"	28" x 48"	28" x 48"	45" x 60"	45" x 60"	45" x 60"	30" x 60"	40" x 60"	32" x 72"	45" x 60"
Dia. of drum.....	10 ft.	Cone	8 ft. 3"	10 ft.	10 ft.	14 ft. 8"	14 ft. 8"	14 ft. 8"	12 ft. 8"	12 ft. 8"	12 ft. 8"	20 ft. 8"
Capacity of tanks, gallons.....	1400	2600	2600	1440	1320	2400	2400	2000	1400	1800	1800	2000
Rated capacity of hoist, 24 hrs.....	2,350,000	3,750,000	3,500,000	1,620,000	1,900,000	3,804,000	3,804,000	3,750,000	2,016,000	2,160,000	2,160,000	5,500,000
Maximum one-day hoist.....	838,814	3,772,600	3,550,000	1,647,000	1,991,880	3,643,787	3,643,787	3,750,000	2,016,000	2,160,000	2,160,000	5,500,000
Average one-day hoist.....	3 years.	2,977,753	2,803,142	1,431,829	1,161,100	Year, 1902.	Year, 1902.	.....	.....	.....	.....	.....
Time of average.....	1890-1901	38 days	38 days	37 days	37 days	.....	.....	.....	.....	.....	.....	.....
1 day record for plant.....		9,077,400 gallons		4,144,680 gals.		7,286,400 gals.						
Quantities in Cubic Feet.												
Capacity of Tanks.....	188	348	348	201	193	322	322	268	188	241	241	268
Rated capacity of hoist, 24 hrs.....	314,900	502,500	469,000	217,000	281,400	569,740	569,740	502,500	270,140	289,440	289,440	737,000
Maximum one-day hoist.....	112,400	505,528	476,900	220,700	303,000	488,270	488,270	502,500	270,140	289,440	289,440	737,000
Average one-day hoist.....		399,020	375,620	191,865	251,730	.....	.....	.....	.....	.....	.....	.....
1 day record for plant.....		1,216,370		555,380		976,380						

*Water-Hoists Under Construction January 1, 1903.*

Location.	Summit Branch Coal Co.				Susq. C. Co. No. 6 Shaft. Glen Lyon Water-Hoist.	Hickory Ridge No. 4 Slope. Union Coal Co.	Scott Shaft No. 2. Union Coal Co.
	Williamstown No. 2 Shaft.						
	No. 1 Water-Hoist.	No. 2 Water-Hoist.	East Coal-Hoist Emergency.	West Coal-Hoist Tanks.			
Hoist.....	1650 ft.	1650 ft.	1650 ft.	1650 ft.	920 ft.	600 ft.	1000 ft. ±
Size engines (pair).....	36" x 60"	48" x 60" ?	36" x 48"	38" x 48"	30" x 72"	Geared 3½ to 1 18" x 36"	Geared 2½ to 1 20" x 48"
Dia. drum.....	Cone 10 to 16 ft.	Cone 12 to 18 ft.	18 ft.	Cone 10 to 16 ft.	10 ft. 10"	10 ft.	9' 6"
Capacity of tanks, gals.	2600	4400 ?	1800 ?	1800 ?	2050	1400	1440
Capacity of tanks, cu. ft.....	348	590	241	241	275	188	193
Rated capacity hoist, 24 hrs, gallons.....	3,000,000	5,250,000 ?	1,800,000	1,800,000	2,900,000	1,250,000	
Rated capacity of hoist, per 24 hrs., cu. ft.....	400,000	700,000	240,000	240,000	390,000 Tanks now used as emer- gency- hoist.	167,500	Shaft  now  sinking.
Shaft now sinking—down 1100 ft.							

of the shafts did considerable damage to the timber; and that the collection of ice in the main shafts, which are invariably down-takes for the ventilation, endangered the men in going up and down in their work. These reasons, with the gradual increase of water beyond the capacity of the plant, led to the abandonment of this method of hoisting, so that now these tanks are used only in emergencies. The method was, however, probably one of the cheapest ever devised for handling a moderate amount of water from deep shafts, as practically the only cost was for the steam used, the extra wear and tear of engines, ropes, shaft-guides and timbering, and the extra oil required for lubrication. The hoisting-engineers being required by the Pennsylvania Mine Law to be in the engine-houses at all times, and night-firemen being necessary at all colliery plants, there is really no additional labor-cost to this method of hoisting. These tanks have a capacity of 1300 gallons (174 cu. ft.) each, and 50 per hour was an ordinary dump, so the total capacity, from a shaft 1000 feet deep, was about 750,000 gallons (8700 cu. ft.) per day of twelve hours.

The present method of hoisting from a special water shaft or water compartment was, I believe, first used in 1896, at the Luke Fidler colliery, Shamokin, by Mr. Morris Williams, then

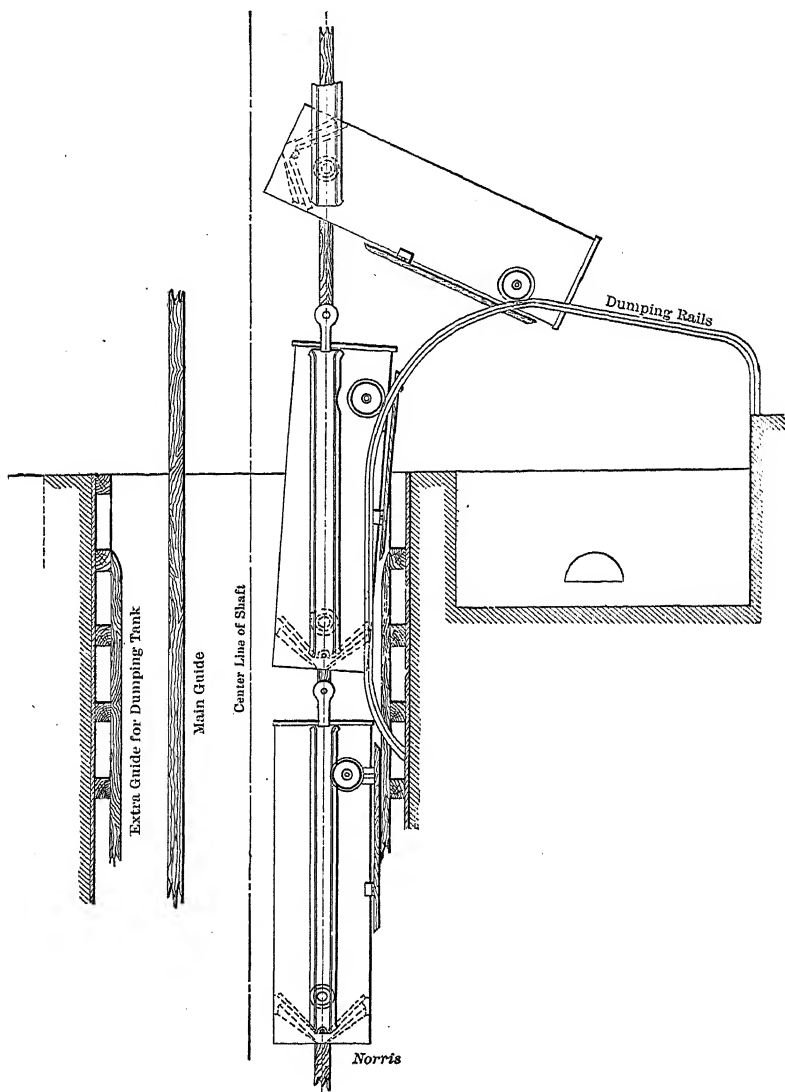


Cage-Tank, Susquehanna Coal Co., Naticoke Collieries, Nos. 5, 6 and 7.



Superintendent of the Mineral Railroad and Mining Co., of which Mr. Irving A. Stearns was the Manager. The plan was

FIG. 2.



Showing Guides and Method of Dumping.

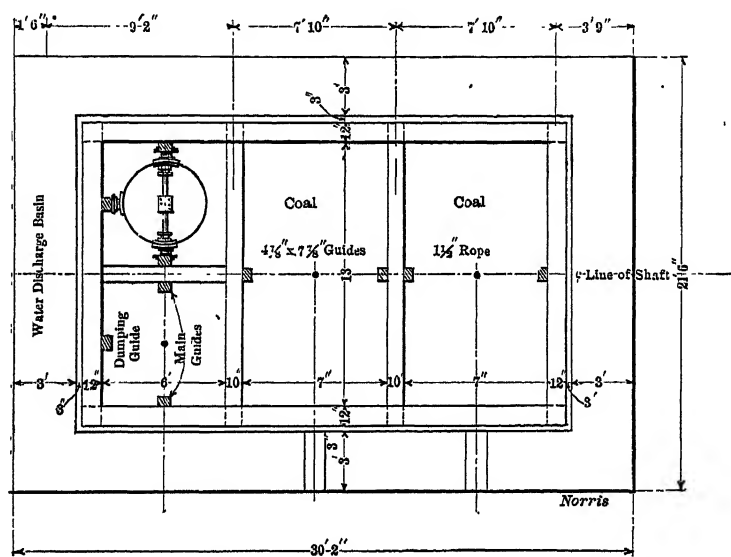
the outcome of the successful use of tanks in unwatering the colliery, which had been flooded to subdue a mine fire.

The tanks were made to dump as shown in Fig. 2, and, to

get the maximum size, were made square with angle-iron corners; it was found almost impossible to keep them tight, and round tanks have been substituted.

The original method of dumping was the use of small wheels bearing against the guides to retain the tanks in a vertical position while hoisting; these passed through slots in the guides when the main dumping-wheels reached the dumping-rails (Fig. 2). These small wheels, if made of iron, rapidly destroyed the guides, and if of softer material only lasted for a few hoists.

FIG. 3.

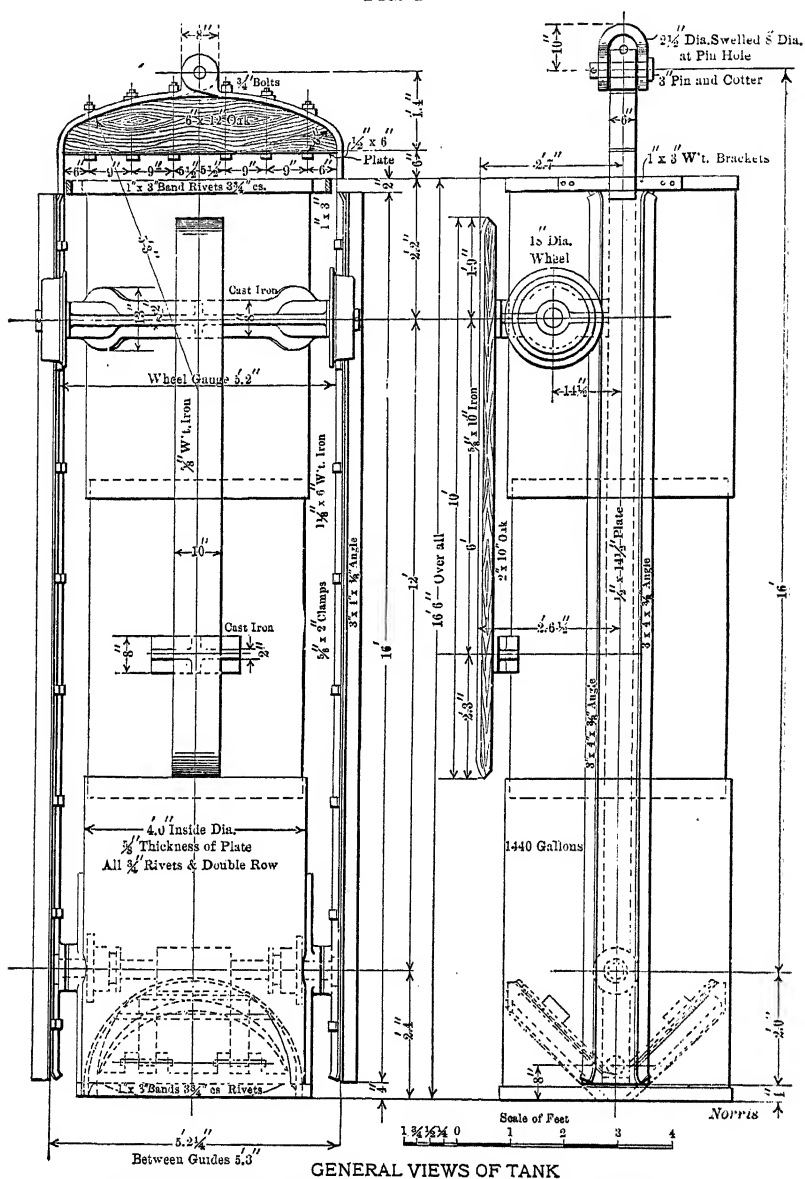


Plan of the William Penn Shaft, Susquehanna Coal Co., showing a 1440-Gallon Tank, Timber and Guides.

The present method of handling these dumping-tanks is the use of a third guide (Figs. 2 and 3) at right-angles to the main guides, and providing the tank (Fig. 4) with an extra shoe, set so as to give the tank a slight tilt toward the third guide; this shoe slides on the guide, keeps the tank steady and in a vertical position until the dumping-wheels, near the top of the tank, engage the dumping-rail at the top of the shaft, and the tank turns gently, pouring its contents into the discharge-basin. Another great advantage of the third guide is the steadiness it imparts to the tank and the smaller liability to accident from shaking the guides loose.

The original tanks were provided with a single, large, flat, clack-valve at the bottom, which, in dumping, tended to swing

FIG. 4.

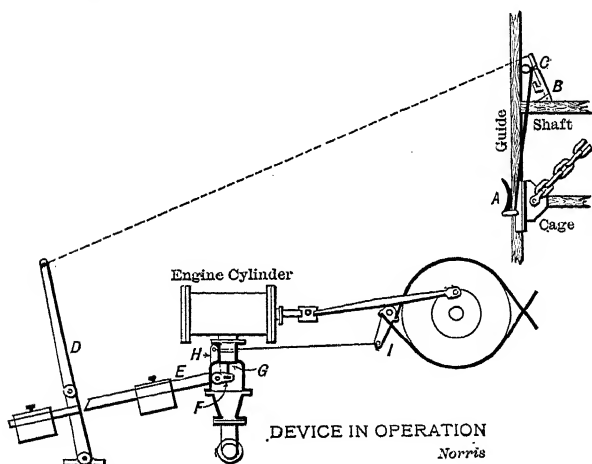
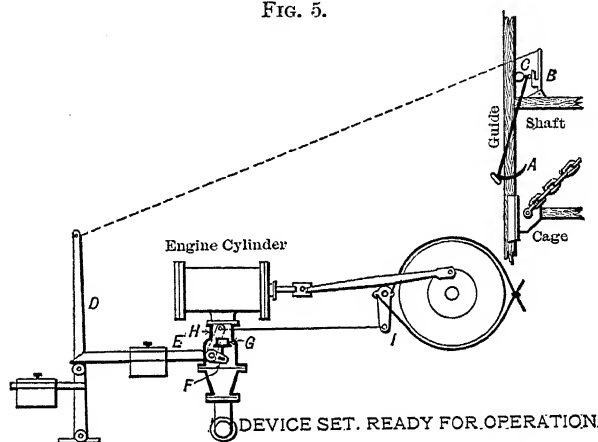


Water-Hoist Dumping-Tank of the William Penn Colliery.

to a vertical position and allow a small escape of water through it; this also struck the water very heavily in descending. The

latest tanks are constructed with butterfly-valves, set on a forty-five-degree angle. This arrangement entirely obviates the loss in dumping, and allows the tanks to enter the water as a wedge with a much less severe shock (Figs. 4 and 8 to 11).

FIG. 5.



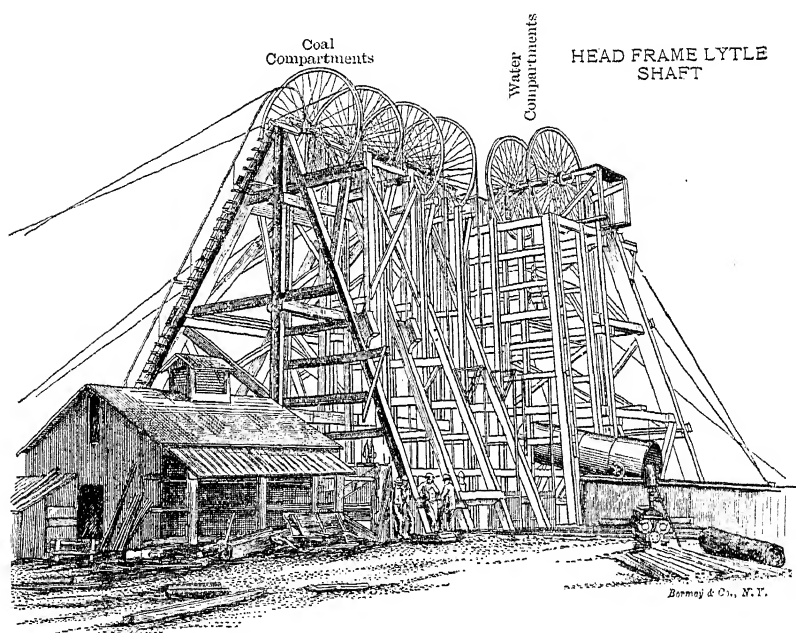
Kohlbraker and Williams Automatic Overwinding Device for Hoisting-Engines.

To successfully operate these dumping-tanks, it is essential to arrange a rest in the sump on which the descending tank is supported while the upper tank is dumping. Without this the lower tank and rope may overbalance the upper tank, with nearly half its weight supported on the dumping-track, and raise it sufficiently to reverse it in the shaft, and possibly do damage to the sheave.

The sudden reduction of load by discharging the water makes the landing a little delicate, and constitutes practically the only objection to this method of hoisting.

An automatic device is usually employed on shafts to minimize the danger from overwinds when hoisting water. The type used by the Philadelphia and Reading Coal and Iron Company and the Pennsylvania R.R. Co. is known as the Kohlbraker & Williams overwinding device. It consists (Fig. 5) essentially of cut-off valves, G, close to the engine-cylinders and

FIG. 6.



a trip-lever, A-B-C, in the shaft operated by the cage or tank, which, when struck, drops the weighted arm, E, and by the movement of the lever, D, instantly cuts off the steam at the cylinders by the closing of the valves, G, and applies the brake through the arm, H. This will absolutely prevent damage from overwinding if the engine is started in the wrong direction or the landing is miscalculated, and will minimize the damage caused by a runaway overwind, which, with the enormous weights and high speeds involved, could not be stopped short by anything less than the absolute wrecking of the engine. In two recent cases the damage from such overwinds was lim-

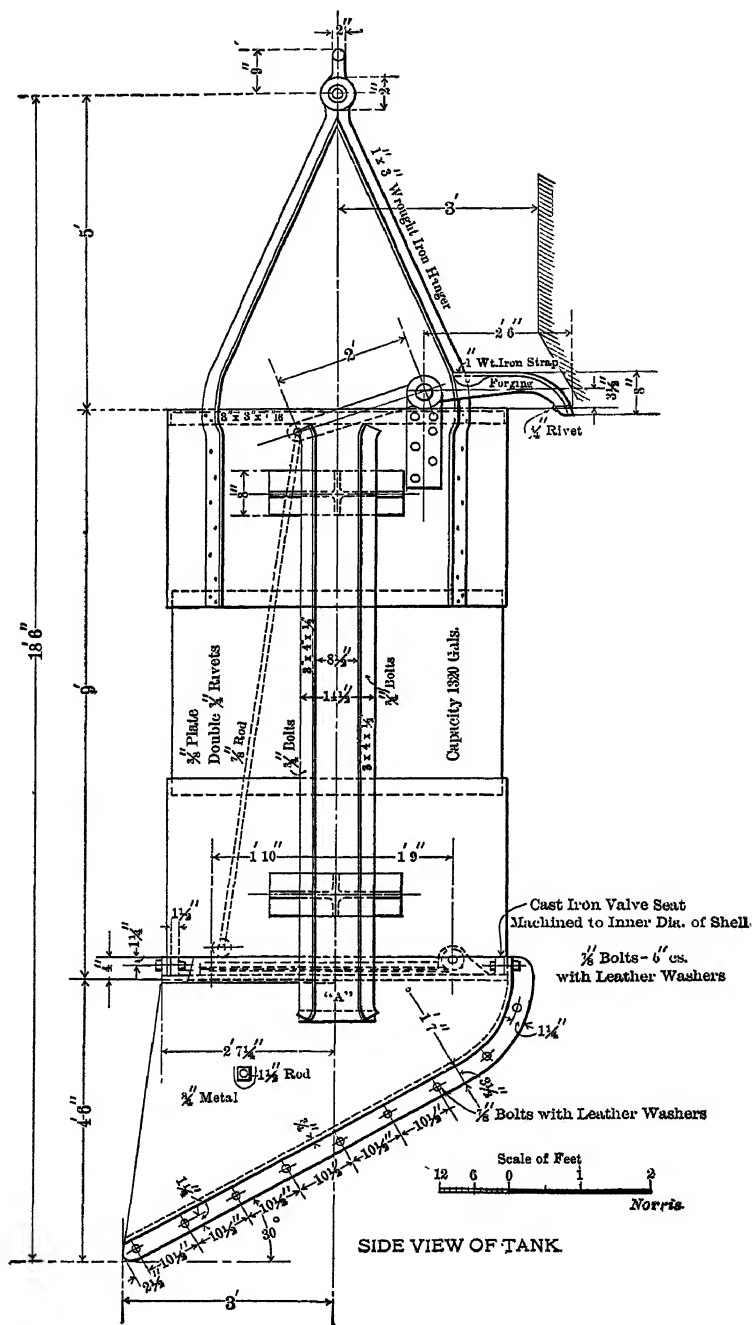
ited to the wreck of the sheave and tank, this result being attained by the use of a head-frame of the type shown (Fig. 6), in connection with the overwinding device. This type of frame gives a clear-through opening, with only the sheave in the line of the hoist.

As the usual dimensions of shaft-compartments are about 7 x 13 feet, it is the general custom to use only one compartment for water-hoisting (Fig. 3). This compartment is divided into two parts by an extra line of buntons (when discharging at the end of the shaft), or two sets of guides are put on the sides of the compartment, with the extra guides on the ends when dumping at the sides of the shaft. The arrangement of two tanks in one compartment also reduces to a minimum the extra size and cost of shaft required for water-hoisting. The water-hoisting engines are then usually set at right-angles to the coal-engines, to avoid placing one sheave over the other, with the resulting extra liability to wrecks.

Bottom-dump tanks, instead of end-dumping ones, are exclusively used by the Philadelphia and Reading Coal and Iron Company; and also by our companies, as emergency-hoists in the coal compartments. These are generally constructed (Fig. 7) with the intake-valve at the bottom, and are provided with a trip-lever, operated by a guide in the head-frame, to raise this valve at the top, and a discharge-casting to direct the outflowing water to one side into basins or troughs. The various types of discharge-castings and valves in use are shown in Figs. 8 to 11; in all of these it will be noticed that an effort has been made to reduce the blow (incident to striking the water) by the use of wedge-shaped castings; the Lytle Coal Co., by the use of a wedge-shaped sheet-iron shield, outside of the discharge-casting, reduces the shock still more.

The objections to this type of tank are: its unsteadiness in hoisting at high speed; slower discharge through the bottom valve (the experience at William Penn colliery having shown an advantage of 10.1 per cent. in favor of the end dump), and, greatest of all, the danger of damage to the guides caused by the slanting nose striking the water and the consequent side-pressure on the tank, which is intensified by the reaction of the water entering on one side only. In our experience with both types there has been practical immunity from trouble with the guides from the end-dump hoists, and almost constant diffi-

FIG. 7.

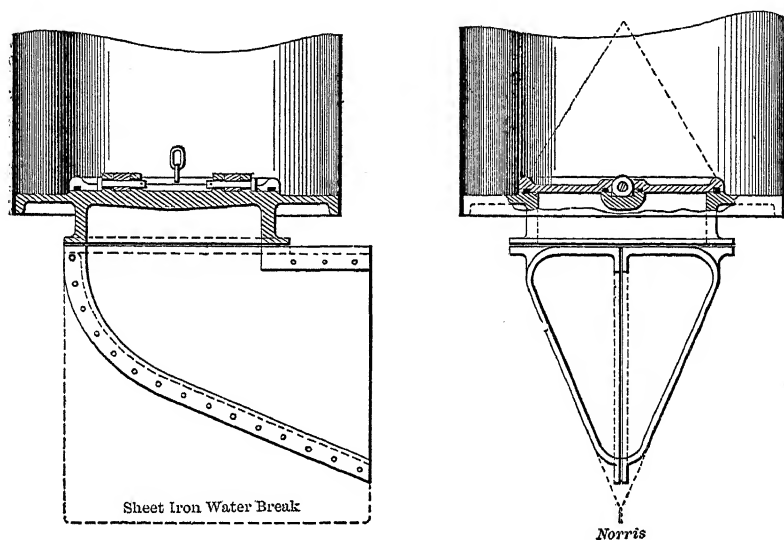


Emergency-Tank of the William Penn Colliery.

culty with them when the bottom-dump tanks are used. One guide at the water-level, forced out after thirty-days' service, was cut into fully  $1\frac{1}{2}$  inches by the wear of the tank-guides; the cast bottoms are also more liable to breakage from striking obstructions or floating timber than are the wrought-iron tanks, the cast valve-seats of which can be made heavy enough to withstand any ordinary battering without unduly increasing the weight of the tanks.

The discharge-casting (Fig. 10) is designed to minimize these

FIG. 8.



Lytle Coal Co.'s Bottom-Discharge, 5-ft. Tank.

difficulties; the idea was suggested to the writer by Mr. George Hill, Cons. Elect. Eng'r for the Delaware, Lackawanna and Western R.R. Co. The plan is to make an open casting, which is a perfect wedge provided with a central partition, to take water in on both sides, and thus entirely avoid the side thrust due to the slanting bottom and side entry of the water. In discharging, arrangement may be made for taking care of the double discharge, or, where this is undesirable, only one valve may be opened, making the discharge in all respects similar to that at present used. To compensate for the disadvantages above mentioned, the bottom-dump tanks do not require any special shaft or dumping preparation; the costly complication of mounting them on trunnions with a through shaft and

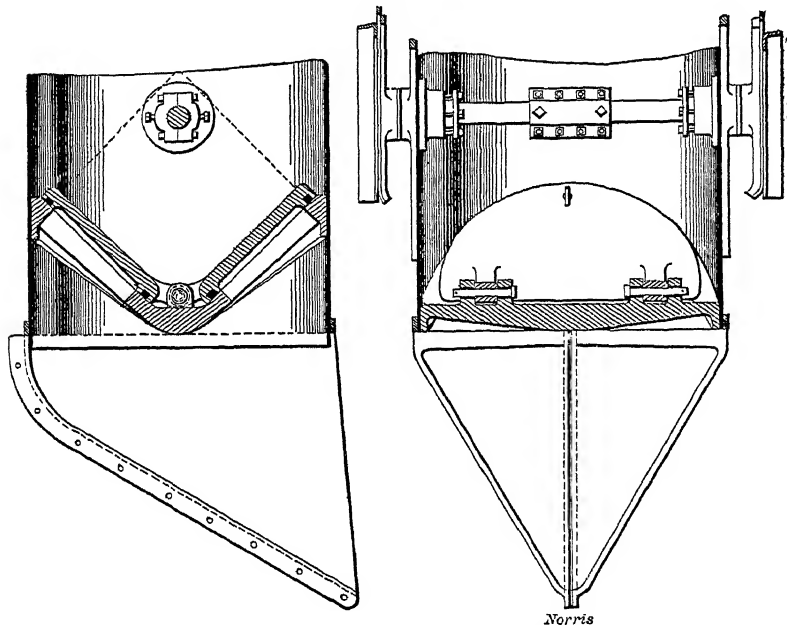


stuffing-boxes is avoided; and the difficulty of landing, due to the sudden change of load, as before mentioned, is much reduced.

The time required for emptying bottom-dump tanks averages about 8 seconds, while the actual stop for end-dumps is but about 2 seconds, though some of this advantage is lost by the necessarily slower landing of the latter.

While all the regular water-hoists are in shafts, very large quantities of water have been hoisted from slopes in emergencies. The tanks are usually of the end-dump type, generally as

FIG. 9.



Susquehanna Coal Co.'s 5-ft Tank with Bottom-Discharge Casting. The same tank is also used for top-discharge without this casting.

described by Mr. J. H. Bowden (*Trans.*, xx., 343), though not used tandem, and have done excellent work. The principal objections to their regular employment are: the rapid wear of the wheels caused by acid mine-water working into the bearings and replacing the oil; the slower hoisting-speed necessary for tanks running on wheels, as compared with those in shafts sliding on guides; the liability to derailment at any point of the hoist; the extreme danger of derailment when entering the water; and the danger, on flat slopes, of obstructions remaining on the rails under water.

There is, however, now being erected at the Hickory Ridge colliery of the Union Coal Company, Shamokin, a permanent water-hoist on a 70-degree slope (Fig. 12), in which it is believed that many of these difficulties have been obviated. The tanks have a capacity of 1400 gallons each, and are mounted on closed, self-oiling wheels with bronze bushings, and close-fitting bronze shields in the end of the hubs, which fit over bronze collars on the axles. The wheels are made with extra high flanges, and the tank is provided with top- and side-shoes, as shown, which slide in between permanent guides, at the foot of the slope, extending 20 feet above the water-line, to avoid danger of derailment when striking the water; the pitch being 70 degrees, there will be no danger of obstructions remaining on the rails under water.

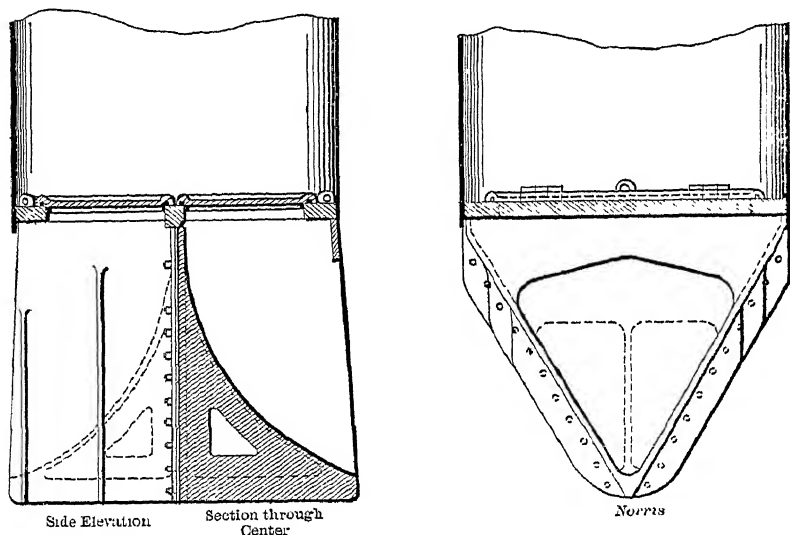
The cost of construction of only two recent plants is available. In this cost the water-hoisting plant is charged with its proper proportion of the total cost of the shaft-sinking, head-frames, steam lines and boiler-plant.

	William Penn Water-Hoist.	Lytle Water-Hoist.
Depth of shaft.....	953 ft.	1500 ft.
Capacity of tanks.....	1440 gals.	2600 gals.
Size of engines.....	32 x 48 in.	36 x 60 in.
Size of drums.....	Straight 12 ft. diam.	Cone 10 to 16 ft. diam.
Capacity of hoist, 24 hours.....	2,100,000 gals. (280,000 cu. ft.)	3,750,000 gals. (500,000 cu. ft.)
Best record, 24 hours.....	2,291,040 gals. (307,000 cu. ft.)	3,772,600 gals. (505,500 cu. ft.)
<i>Cost:</i>		
Sinking and timbering.....	\$20,673.81	\$22,641.63
Head frame.....	4,224.13	3,540.58
Water-hoist engines, foundations and house.....	15,583.64	29,653.17
Tanks and ropes.....	2,393.23	3,899.65
Steam line.....	3,726.12	4,951.17
Boiler-plant.....	.....	16,091.76
	\$46,600.93	\$80,777.96
Cost, excluding shaft sinking and steam-plant.....	\$22,201.00	\$37,093.40
Cost per 1000 gals. daily capacity, excluding shaft and steam-plant.	\$10.57	\$9.87
Cost per 1000 cu. ft. daily capacity, excluding shaft and steam-plant.	\$88.08.	\$82.25

The cost of the steam-plant is omitted from the lower set of figures because it was available in only one case, and in that

was based upon a division of cost among three hoists, in proportion to size of engines; the cost of steam lines depends on the location of the boilers in relation to the shaft, and the distance in both cases is considerable. The cost of sinking is independent of the capacity of the plant, as the William Penn 1440-gallon (193 cu. ft.), the Lytle 2600-gallon (348 cu. ft.), and the proposed Williamstown 4400-gallon (590 cu. ft.) tanks are all in compartments of practically the same size. The greater cost of the William Penn plant is fully accounted for by the rise in the price of materials during the past three years and the decreased efficiency of labor in this region.

FIG. 10.



Susquehanna Coal Co.'s 5-ft. Tank with Double Bottom-Discharge Casting.

The three water-hoists on which costs of operation are available are at the Luke Fidler, Lytle and William Penn shafts; of these the William Penn was first started, October 30, 1902, to unwater the colliery after the strike, the plant having been finished during the trouble; the Lytle has been in operation for about two years, but, owing to the uncompleted condition of the colliery, its work has been irregular, except in removing the water which accumulated during the strike; while at the old Luke Fidler plant the records are in excellent condition, excepting for the steam consumption, which are unavailable.

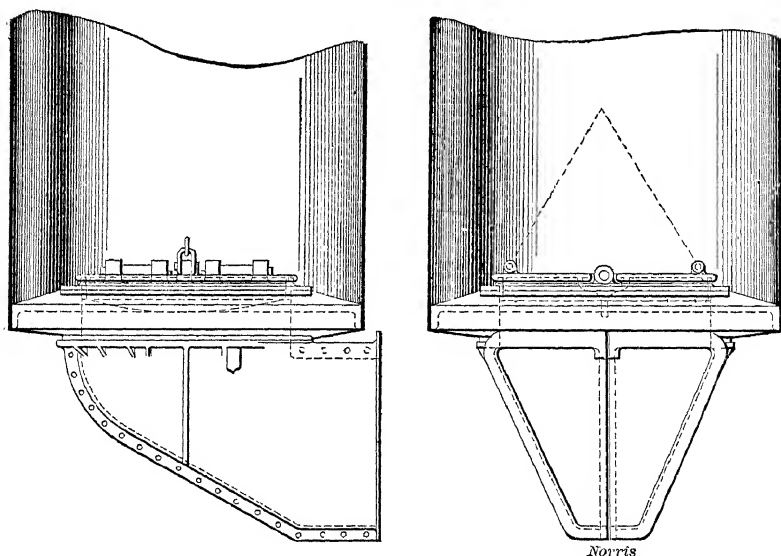
The Lytle shaft was filled to a depth of 860 feet during

the strike and the water, amounting to 274,083,500 gallons (36,727,189 cu. ft.), was hoisted out in thirty-seven days and four hours. Besides the regular water-hoist, tanks were used in all four coal compartments; the plant then consisting of two pairs of 2600-gallon (348 cu. ft.) tanks, with a pair of 36 x 60 in. first-motion engines for each pair; also one pair of 1500-gallon (201 cu. ft.) tanks operated by a pair of 30 x 48 in. Corliss engines; the total water hoisted by each was:

	Water Hoisted.		Average Per Day.	
	Gallons	Cu. Ft.	Gallons.	Cu. Ft.
Water-hoist.....	113,154,600	15,162,716	2,977,753	399,019
Large coal-hoist.....	106,519,400	14,273,600	2,803,142	375,621
Small coal-hoist.....	54,409,500	7,290,873	1,431,829	191,865
Total.....	274,083,500	36,727,189	7,212,724	965,505

The hoisting during the month of November was quite regular; 236,906,000 gallons (31,745,300 cu. ft.) were hoisted from an average depth of 740.6 feet, which is equivalent to 1,463,572,816,468 ft.-pounds. During this time the boiler-plant (12-150 H.-P. return tubular boilers and one 500 H.-P. B. & W. boiler) was devoted exclusively to this hoisting, burned 4122 tons

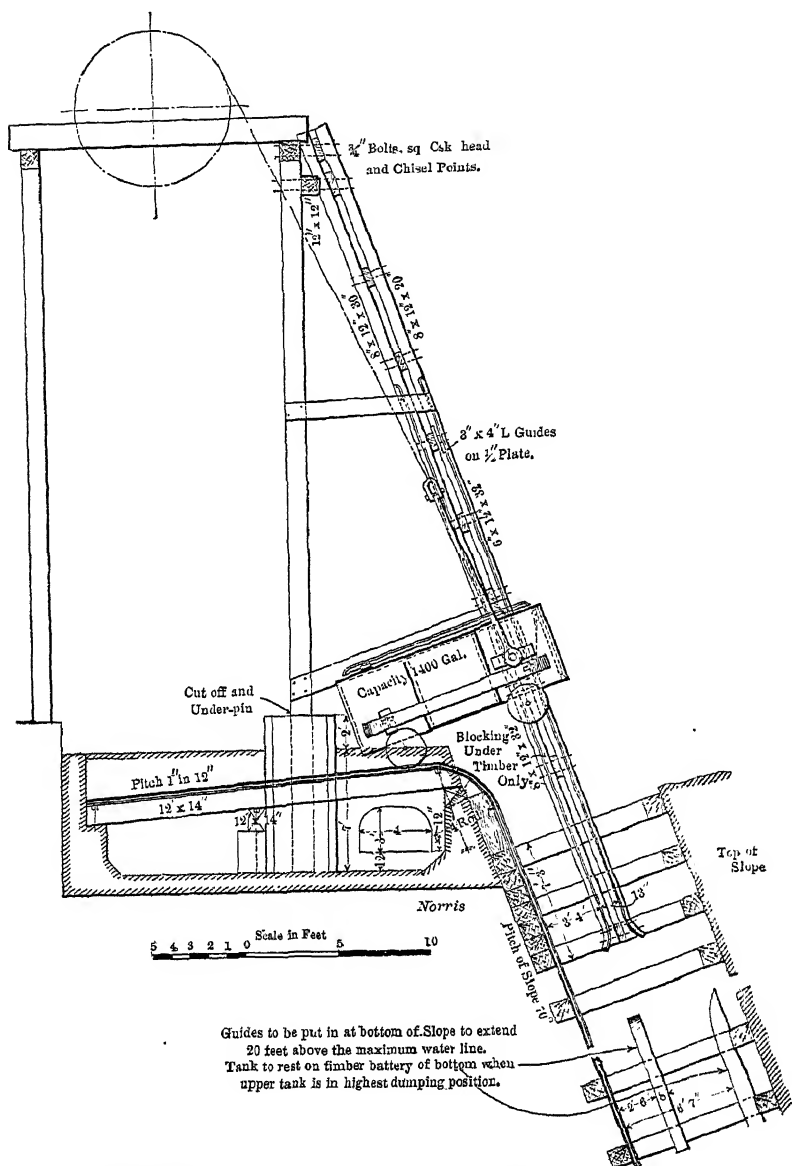
FIG. 11.



Philadelphia and Reading Coal and Iron Company's Bottom-Discharge  
5-ft. 6-in. Tank.

of coal, and was supplied with 6,206,100 gallons of metered water, which would indicate an average evaporation of 5.55 lbs. per

FIG. 12.



Water-Hoist of Union Coal Co.'s Hickory Ridge Slope No. 4.

pound of coal. Allowing 15 per cent. of the water supplied for use in blowers, condensation and waste, the amount of dry

steam at the engines would be about 44,004,000 lbs. This would give an approximate duty of 33,260,000 ft.-lbs. per 1000 lbs. of dry steam; or,  $59\frac{1}{2}$  lbs. of steam per actual horse-power hour in water lifted, and 251 lbs. of steam per 1000 gallons lifted 1000 feet (1878 lbs. steam per 1000 cu. ft. water lifted 1000 ft., or about 30 lbs. steam per 1,000,000 ft.-lbs. in water). This is based on the assumption that the steam used would be proportional to the hoist, an assumption which would be against, rather than in favor of, the plant.

The cost of steam during this month was as follows:

Labor, repairs and supplies, . . . . .	\$934.62
Water, . . . . .	496.49
4122 tons coal at 50 cents per ton, . . . . .	2061.00
	<hr/>
	\$3492.11

Thus about 44,004,000 lbs. of dry steam delivered at the engines cost \$0.0794 per 1000 lbs. (labor, repairs and supplies, \$0.0326; fuel, \$0.0468); equivalent to \$0.0198 per 1000 gallons hoisted 1000 ft. vertically (\$0.1485 per 1000 cu. ft.); or \$0.00238 per 1,000,000 ft.-lbs. in water; \$0.00472 per H.-P. hour in water; cost of steam only, per year, per boiler H.-P., 24 hrs. per day, including labor, supplies, and repairs, \$8.57; fuel, \$12.30; total, \$20.87. The cost for generating steam at the collieries of the Lykens Valley Co., figured on the same basis, from data as given in another paper, allowing 15 per cent. waste as above, is \$0.0811 per 1000 lbs. dry steam delivered at pumps (\$0.0389 for labor, repairs and supplies, and \$0.0422 for fuel), and averages \$17.04 per year per boiler H.-P. 24 hours per day.

The cost of the steam supplied has been given in detail in this place because this plant is the only one at which a single boiler-plant has been devoted to water-hoisting for a sufficient time to make possible a fairly accurate estimate of the cost of power, and the figure here obtained, 2 cents per 1000 gallons (15 cents per 1000 cu. ft.) raised 1000 ft. vertically, will therefore be used throughout this paper. The cost of hoisting, exclusive of steam, during this month was \$1676.74, or \$0.0071 per 1000 gallons (\$0.053 per 1000 cu. ft.) hoisted, including all supplies, labor and repairs.

The new plant of the William Penn No. 2 shaft, which was flooded to a depth of 250 feet, hoisted 112,468,080 gallons (15,070,730 cu. ft.) of water from October 30 to December 5,

1902, inclusive, using the pair of regular water-hoist, 32 x 48 in. engines, and the pair of 28 x 48 in. coal engines, with 1440-gallon (193 cu. ft.) end-dump tanks (Fig. 4), and 1320-gallon (177 cu. ft.) emergency bottom-dump tanks, Fig. 7, the record being:

	Water-Hoist.	Emergency Hoist.	Total.
Tanks hoisted, . . .	48,269	32,546	80,815
Hours actually worked, .	805½	598½	1403½
Tanks per hour, . . .	59.9	54.4	
Gallons hoisted, . . .	69,507,360	42,960,720	112,468,080
Cu. ft. " . . .	9,313,985	5,756,745	15,070,630
Average height hoisted, . . .			727.8 ft.

The total cost, exclusive of steam, was \$987.83, or \$0.0088 per 1000 gallons actually hoisted (\$0.066 per 1000 cu. ft.).

The following is the record of the Luke Fidler shaft water-hoist 32 x 48 in. engines, with 1450-gallon (188 cu. ft.) tanks, for the three years of regular operation, 1899, 1900 and 1901:

Height of hoist, . . . . .	960 feet.
No. of tanks hoisted, . . . . .	633,456
Average No. per year, . . . . .	211,152
Gallons hoisted, . . . . .	918,501,200 = 123,079,160 cu. ft.
Average per year (gallons), . . . . .	306,167,067 = 41,026,386 cu. ft.

		Per 1000 gallons.	Per 1000 cu. ft.
Cost of labor, . . . . .	\$6,372.00	\$0.00695	\$0.05212
Supplies and repairs, in- cluding two new ropes, one sheave and one tank. }	4,087.20	0.00445	0.03337
Total, exclusive of steam, . . . . .	\$10,459.20	\$0.0114	\$0.0855
Steam on basis of Lytle figures. }	17,635.22	0.0192	0.1440
Total cost, . . . . .	\$28,094.42	\$0.0306	\$0.2295

This plant was, however, only operated at about one-third of its capacity; at full capacity the cost is estimated to average about two and one-half cents per 1000 gallons for 960 ft. vertical.

The summary of the operating costs of the three plants just referred to is shown in the tables on page 125.

This is much less than (about 69 per cent. of) the average cost of pumping at the collieries of the Lykens Valley Coal Co., where it was \$0.37 and \$0.29 per 1000 cu. ft. 1000 ft. vertical, and \$98.11 and \$81.47 per H.-P. year in water for the years 1901

Plant.	Fidler.	Wm. Penn.	Lytle.
Time.	3 Years.	37 Days.	1 Month.
Depth of shaft.....	960 ft.	953 ft.	1500 ft.
Quantity hoisted, gallons.....	918,501,200	112,468,080	236,906,000
Quantity hoisted, cu. ft.....	123,079,160	15,070,730	31,745,300
Average height hoisted.....	960 ft.	727.8 ft.	740.6 ft.
Cost of labor, repairs and supplies per 1000 gallons.....	\$0.0114	\$0.0088	\$0.0071
Cost of steam per 1000 gallons.....	0.0192	0.0146	0.0148
Total cost per 1000 gallons.....	\$0.0306	\$0.0234	\$0.0219
Total cost per 1000 cu. ft.....	\$0.2295	\$0.1755	\$0.1643

Estimated Cost Per 1000 gallons, and 1000 cu. ft., 1000 feet vertical.	Fidler.		Wm. Penn.		Lytle.	
	1000 Gallons.	1000 Cu. ft.	1000 Gallons.	1000 Cu. ft.	1000 Gallons.	1000 Cu. ft.
Labor, supplies and repairs for hoisting.....	\$0.012	\$0.090	\$0.009	\$0.068	\$0.008	\$0.06
Steam.....	0.020	0.150	0.020	0.150	0.020	0.15
Total.....	\$0.032	\$0.240	\$0.029	\$0.218	\$0.028	\$0.21
Total cost per 1,000,000 ft.-lbs. in water.....	\$0.0038		\$0.0035		\$0.0034	
Total cost per H.-P. year, 24 hrs. per day, in water.....	\$65.91		\$60.71		\$58.97	

and 1902, respectively, as shown in my other paper\* (and which it was estimated could be reduced to about 25 cents and 19 cents in the respective years by the use of a modern compound condensing pumping-plant). In this comparison it should also be considered, however, that the steam-cost of hoisting could, if desirable, be proportionately reduced by the use of compound engines, condensing or non-condensing, or even by running the present simple engines as condensing.

Aside from the question of cost, however, there are many very great advantages in hoisting water, particularly from deep shafts: (1) the simplicity of the construction; (2) having all the operating machinery on the surface, with the resulting low cost of repairs, which are practically confined to tanks and ropes; (3) the almost total absence of slip, which under mining conditions reduces materially, from the quantity calculated from

\* "The Cost of Pumping at the Short Mountain Colliery of the Lykens Valley Coal Company," present volume, p. 127.



“plunger displacement,” the actual quantity of water pumped; (4) the avoidance of underground steam lines, with their large condensation losses, damage to roof and timbering from the heat and exhaust steam, and the danger of fire incident to their use; (5) the almost total freedom from danger of falls or squeezes in the mines; and (6), most of all, because the operating plant cannot be flooded.

These advantages were brought home most forcibly to the anthracite operators this year, when, after a six-months' strike, the water-hoist collieries were promptly unwatered, while in one case, at least, those where the pumps had been drowned are still flooded in the lower levels. The history of the Lytle colliery, a notoriously wet one, also furnishes a case in point. Until the accession of the present management, the regular report was, “we are holding the water,” and it required seven months of unremitting effort moving pumps down the slopes, and a tremendous expenditure of money, to clear the colliery of a volume of water barely equal to that removed by the hoisting-plant this fall in thirty-seven days, at an expenditure (excluding the value of the coal burned) of \$5918.63.

While the figures presented in this paper are by no means scientifically accurate, the writer has expended no small amount of time and trouble in getting the very best data available, and he believes that the results are as nearly correct as the circumstances will permit, and hopes the figures may be found of value to the profession.

It is also to be noted that, on account of the strain of this constant work on the hoisting engineers, it is the custom to have them work in three shifts of eight hours each.

In conclusion, the writer desires to acknowledge his indebtedness to Messrs. Arthur Kennedy, E. A. Rhoades and Robert Quinn, Superintendents, respectively, of the Lytle, William Penn and Luke Fidler collieries, for water-hoist data furnished; to Mr. Hood McKay, Superintendent of the Lykens Valley Coal Co., for pumping-data; to Mr. R. C. Luther, General Superintendent Philadelphia and Reading Coal and Iron Co., for data of the Reading water-hoists; and to Mr. Morris Williams, Manager of the Pennsylvania Rail Road Coal Companies, for many valuable points and for permission to publish the detailed costs given.

## The Cost of Pumping at the Short Mountain Colliery of the Lykens Valley Coal Company.

BY R. V. NORRIS, WILKES-BARRE, PA.

(Albany Meeting, February, 1903.)

THE great coal strike of 1902, which confined the work at the Short Mountain colliery of the Lykens Valley Coal Company almost exclusively to pumping, gave an opportunity to determine with considerable accuracy the cost of pumping the water from these very old, extensive and wet mines.

This colliery, which is located in the extreme southwestern end of the anthracite coal-field, has been operated continuously since 1836, and has shipped 7,747,313 tons of coal. This and the Williamstown colliery are the only operations in Dauphin county.

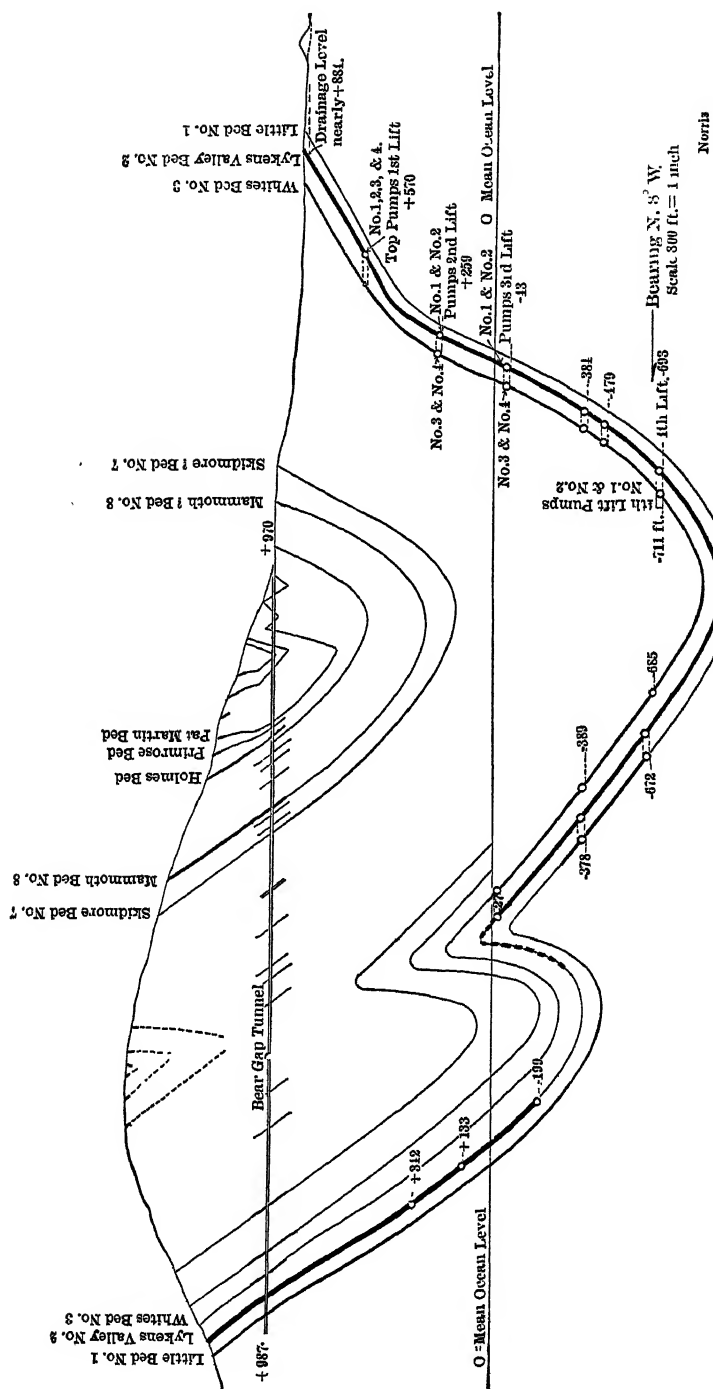
The workings cover about 5 miles along the North dip of the extreme westerly end of the basin, with the operating slopes near the center. They extend around the westerly spoon of the basin and back about  $3\frac{1}{4}$  miles easterly on the South dip. The mines are very deep, the present workings extending to 711 ft. below sea-level, and about 1600 ft. below the lowest surface-opening.

The coal is hoisted principally from the Short Mountain slope, which is 1550 ft. long, with 26 to 68 degrees dip, and from an inside slope, No. 4, 800 ft. long, with 52 to 72 degrees dip. The ordinary capacity of the colliery is about 1000 tons per day, and the average output for the years 1897 to 1901, inclusive, was 297,000 tons per year.

The seams worked are the lowest Red Ash measures, being, respectively, Nos. I, II and III of the coal-bed series, but locally called the "Little Lykens," "Lykens," and "Whites." On the accompanying plan the Lykens seam is shown in full and the Whites in dotted lines.

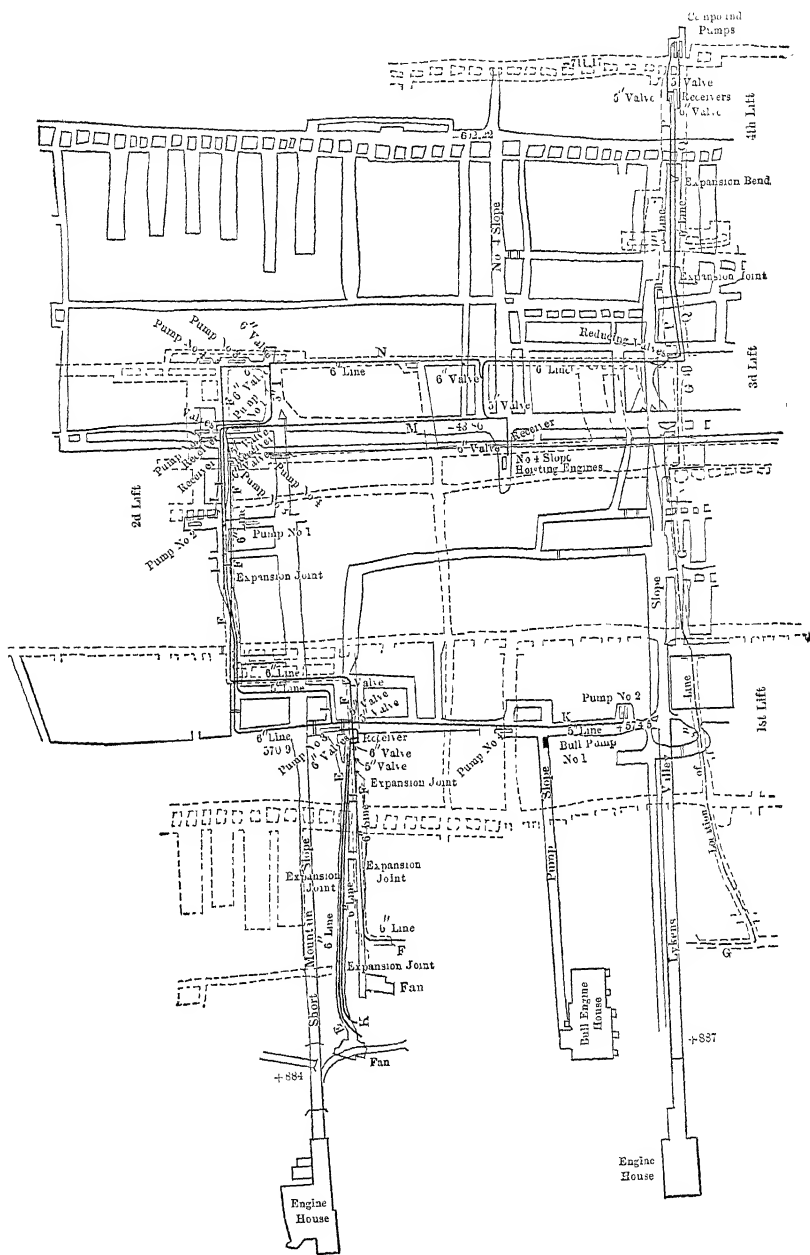
The pumping-plant is divided into four lifts, the pumps being located as shown in the section, Fig. 1, and the plan, Fig. 2.

FIG. 1.



Section (looking east) through Lykens Valley Slope and Bear Gap Tunnel, Lykens Valley Coal Company,

FIG. 2.



Plan of the Short Mountain Colliery, showing location of Pumps and inside Steam-Lines.

The greater part of the water is caught on the No. 3 level and pumped from there to the surface. The pumps on the No. 4 level handle, on an average, only one-third of the total water pumped to the surface. Excepting the bottom lift, the pumps are all simple and direct-acting, and many of them have been in place for a number of years. This plant, while not claiming to be modern or very economical, is a very fair sample of the average plant in use in the great majority of the older and deeper slope-colleries in the anthracite region. In fact, it is, if anything, rather above the average.

The pumps are as follows:

Level.	No.	Maker.	Size.			Size Column. Inches.	Vertical Lift Including Suction Feet.	Capacity. Gallons Per Min.
			Steam. Inches.	Water. Inch.	Stroke. Inches.			
4	1	Jeanesville Comp. Duplex.	24 and 38	10	36	10	673	857
4	2	Jeanesville Comp. Duplex.	25½ and 38	10	36	10	660	857
3	1	Allison & Bannan. Single.	27	12½	108	8	321	692
3	2	Allison. Single.	27	12½	72	10	321	737
3	3	" "	38	16	72	12	326	1086
3	4	" "	30	14½	72	10	326	1156
2	1	" "	36	16½	72	12	331	1131
2	2	Allison & Bannan. Single.	25	12½	72	8	331	658
2	3	Allison. Single.	38	16½	72	12	328	1131
2	4	" "	30	12½	72	8	328	897
1	1	Bull Pump.	44	21½	116	20	319	866
1	2	Griscom Duplex.	22	9½	36	10 and 12	318	1032
1	3	Carter & Allen.	26	11½	72	8	293	852
1	4	Allison.	38	16½	72	14	317	1005

The boiler-plants were tested in May, 1898,\* and the evaporation per pound of fuel (Nos. 2 and 3 buckwheat coal) was found to be: For cylinder boilers, 5.51 pounds, and for Babcock & Wilcox boilers, 6.89 pounds water at actual temperatures and pressures, equivalent to 7.75 and 9.22 pounds, respectively, from and at 212 degrees per pound of combustible. These tests are used here as a basis for deducing the quantity of steam used by the plant as a whole.

\* See my paper on "The Generation and Utilization of Steam by the Lykens Valley Coal Company and Summit Branch Coal Company, Dauphin County, Pa.," *Trans. Am. Soc. Mech. Eng.*, vol. xx., p. 50.

The records of the actual water pumped (plunger displacement) are accurately kept by counters on each pump; the labor-cost of pumping, and the cost of repairs and supplies to pumps are also known. At the boiler-plants the labor-, repair- and supply-accounts and the total coal used for steam at the colliery are accurate; but there has been, until recently, no practical way of dividing the total steam cost between pumping and the other manifold colliery uses.

During June, July and August, 1902, practically all of the steam generated at the colliery was used in pumping. During this time 7692 tons of coal were used for firing, of which it is estimated that 232 tons were used in supplying steam for accommodation-hoisting, ventilation and in condensation in unused steam-lines, leaving 7360 tons for generating steam for pumping. During these three months, 207,034,324 gallons (27,742,600 cu. ft.) were pumped from an average depth of 1152 ft. making an average of 0.035 ton of coal per 1000 gallons (0.277 tons per 1000 cu. ft.). On this basis, correcting for average depth and for use of different proportions of cylinder and Babcock & Wilcox boilers, we find for the years 1901 and 1902 as follows (1901 being corrected for 7.8 per cent. less efficient boiler-plant and 1 per cent. less depth, and 1902 for 2.6 per cent. less efficient plant and 5.3 per cent. less depth):

	1901.	1902.
Total water pumped (plunger displacement), . . .	567,113,616 gallons. (75,990,000 cu. ft.)	1,116,320,253 gallons. (149,587,000 cu. ft.)
Average depth pumped, . . .	1,141 feet.	1,093 feet.
Total estimated coal used, . . .	21,200 tons.	37,963 tons.
Lbs. coal per million ft.-lbs. in water, . . . . .	8.87 lbs.	8.51 lbs.
Lbs. coal per H.P. hour in water,	17.56 "	16.85 "

This estimate is checked by the total coal used for firing: For 1901, 41,288 tons, and for 1902, 49,149 tons; the difference being, for steam uses other than pumping, for 1901 (12 months' work), 20,088 tons, and for 1902 ( $6\frac{3}{4}$  months' average work 6 months full and  $1\frac{1}{2}$  months at about  $\frac{1}{2}$  capacity), 11,186 tons, which is equivalent to 19,886 tons for 12 months.

As the average evaporation of the plant, with the proportion of cylinder and water-tube boilers in use during June, July and August, 1902, was 6.64 pounds of water per pound of fuel,

based on the tests of 1898 above referred to, the total steam made during these three months was about 109,470,000 pounds.

The foot-pounds of work used in pumping were 1,987,529,500,000; the duty of the pumps was about 18,156,000 foot-pounds per 1000 pounds of steam made by the boilers, which should be increased by at least 15 per cent. for steam used in Argand blowers and condensation in steam-lines, giving as the approximate duty of the pumping-plant 20,880,000 foot-pounds per 1000 pounds of dry steam delivered at the pumps.

Dividing the total cost of making steam between the colliery and the pumping-plant in proportion to the coal used, namely, 51 per cent. in 1901 and 77 per cent. in 1902, the total cost of pumping was as follows:

	1901.	1902.
Total cost of labor, supplies and repairs for generating steam, .	\$22,059.72	\$19,728.28
Per cent. used for pumping, .	51 per cent.	77 per cent.
Cost of labor, supplies and repairs in generating steam for pumping only, . . . . .	\$11,250.46	\$15,190.78
Coal used, 21,200 tons, at 50 cents per ton, . . . . .	10,600.00	
Coal used, 37,963 tons, at 50 cents per ton, . . . . .		18,981.50
Total cost of steam for pumping, .	\$21,850.46	\$34,172.28
Cost of labor, supplies and repairs for pumping-plant, . . . .	8,915.06	12,236.09
Total cost of pumping, . . . .	\$30,765.52	\$46,408.37
Total cost per 1000 gallons, . .	\$ 0.0543	\$ 0.0416
Average vertical lift, . . . .	1141 ft.	1093 ft.
Cost per 1000 gallons, 1000 ft. vertical height, . . . .	\$ 0.0495	\$ 0.039
Cost per 1000 cu. ft., 1000 ft. vert.,	0.3712	0.292
" " 1,000,000 ft.-lbs. in water,	0.0060	0.0047
" " H.P. hour in water, . .	0.0112	0.0093
" " H.P. year, 24 hrs. per day, in water pumped, . .	\$98.11	\$81.47
Cost steam only per year per boiler H.H., 24 hrs. per day,	\$17.77	\$16.30

The last results are based on the assumption that the steam used will vary directly as the lift, and that the labor-cost of pumping will not be affected by the slight change in average lift.

While no claim is made for anything like scientific accuracy in the foregoing figures, I am satisfied that they are a very close

approximation to the truth, and the method of arriving at them is given so fully that the probable amount of error, which can only be in the division of the steam account, can be readily seen; further, only those familiar with anthracite work can imagine the difficulty of getting information even as accurate as this. In seventeen years' practice I have been unable, until now, to arrive at even an approximate cost for pumping of this character and extent.

The object of this investigation was to determine the probable saving in operating expenses, in replacing this plant by a more modern one with compound condensing-pumps and higher lifts, which it is now figured will reduce the cost of pumping to about three cents per 1000 gallons (for 1000 feet vertical lift) in dry years like 1901, and to two and two-tenths cents in flood years as 1902, at a cost of about \$120,000.00.

The average amount of water pumped at this colliery ranges from 2000 to 4000 gallons ( $7\frac{1}{2}$  to 15 tons) per ton of production, and the maximum in wet months has reached as high as 8000 gallons or thirty tons of water for each ton of coal shipped from the colliery; and, for the flood year, 1902, when the production was cut down by the strike, reached 8940 gallons, or  $33\frac{1}{3}$  tons for each ton of coal produced. The coal used for steam purposes is in average years about 15 per cent., and in 1902 was 40 per cent. of the shipments of the colliery. In the Williamstown colliery of the Summit Branch Mining Co. adjoining, the coal used for firing was, in 1901, 42 per cent. of the amount shipped, and in 1902 amounted to over  $1\frac{1}{3}$  times the shipments, or 57 per cent. of the total amount mined; it is at this colliery that the large water-hoist shaft referred to in my paper on Water-Hoisting\* is being sunk.

The cost of pumping at the Lehigh Valley Coal Co.'s Hazleton Shaft plant, of high duty, with 5,000,000-gallon pumps, is said to be about  $1\frac{1}{4}$  cents per 1000 gallons for 560-feet vertical lift, and this is almost unquestionably the highest-grade pumping-plant in the Anthracite region.

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\* See page 106 of the present volume.



## Electrical Apparatus for Coal-Mining.

BY W. B. CLARKE, SCHENECTADY, N. Y.

(Albany Meeting, February, 1903.)

It is the purpose of this paper to discuss briefly some of the recent developments in connection with the use of electrical apparatus for the mining of coal. For several years electricity has been employed extensively for the distribution of power, primarily for the operation of mine-locomotives and chain coal-cutters, and secondarily for motors driving pumps, ventilating-fans, rope-haulage, conveyors, crushers, auger-drills, coke-oven larries, etc.

The severe and peculiar conditions which attend the development and distribution of power for the operation of coal-mining machinery constitute a field to which the electric-motor seems to be especially adapted. The underground use of steam-engines, to any great extent, is impracticable for obvious reasons. The much higher efficiency of the electrical system is a potent argument in its favor as compared with compressed air, which, under ordinary circumstances, should only be employed for the operation of rock-drills and coal-"punchers." The conditions in many mines prohibit the under-cutting of coal with a chain-machine; and the compressed-air "puncher," with its very low efficiency, is extensively employed.

There have been no recent important improvements in the design and construction of the chain-machine, although, of course, minor changes are continually being made. More time and attention are being paid now than ever before to the details of mining-machinery, with the idea of eliminating the weaknesses which have been developed, and especially of facilitating repairs and renewals.

Electric mine-locomotives are very extensively employed for the haulage of coal on the main entries and the branch-headings; but in the majority of coal-mines the cars are still

"gathered" from the breasts by mules or horses. In a few low-vein mines, where very small cars are necessarily used, the miners push the cars between the working-faces and the "room-necks," from which points they are collected by locomotives and hauled in trains to the tippie or shaft-bottom. This practice is very limited, and is confined to practically level rooms. Mine-haulage with mules or horses is very expensive and extremely unsatisfactory, especially where large cars are used, or where the rooms are driven on even moderate grades. It would be decidedly advantageous in many mines to employ larger and fewer cars or to have heavier grades in the rooms, provided a suitable substitute for the mule were available. The use of larger cars, handled expeditiously, would materially increase the output of a mine, with the same amount of development. In many veins it is necessary to increase the height along all of the haulage-rooms in order to accommodate the smallest animals obtainable. The cost of "brushing" the roof, or taking up the "bottom," is a formidable expenditure, especially since the material encountered is usually a hard slate or limestone. This would be eliminated in many cases by the adoption of electric mine-locomotives. It is evident from the above remarks that an efficient gathering-locomotive would meet a very important requirement in mine-operation. For service of this nature, compressed-air and electric storage-battery locomotives have been used to some extent. The compressed-air locomotive is not an unqualified success for the following reasons:

1. The radius of operation is restricted. The locomotive must necessarily be very compact, and consequently has small storage-capacity, so that the reservoirs require frequent recharging.

2. The efficiency of the compressed-air locomotive is very low, usually less than 40 per cent., while the plant-efficiency, *i.e.*, the ratio between the effective work performed by the locomotive and the indicated horse-power of the steam-cylinders of the air-compressor, seldom exceeds 20 per cent. In comparison with these figures, the plant-efficiency of an electric mine-haulage system, considering the losses in the engine, generator, line and locomotive, is ordinarily about 55 per cent.

3. Finally, the compressed-air plant costs from two to three times as much as an equivalent electric mine-haulage system. From the lower efficiency, it follows that the boiler-plant must be much larger, while the compressed-air locomotive itself costs much more than an electric locomotive. A three- or four-stage air-compressor is an expensive piece of apparatus, and the heavy pipe-line, incident to the high pressure, is very costly indeed. In this connection it might be well to add that the maintenance of the compressed-air plant, as compared with the electric system, is exceedingly high. Compressed-air locomotives are relatively complicated, and are subject to very frequent repairs, with consequent interruption in the output of the mine.

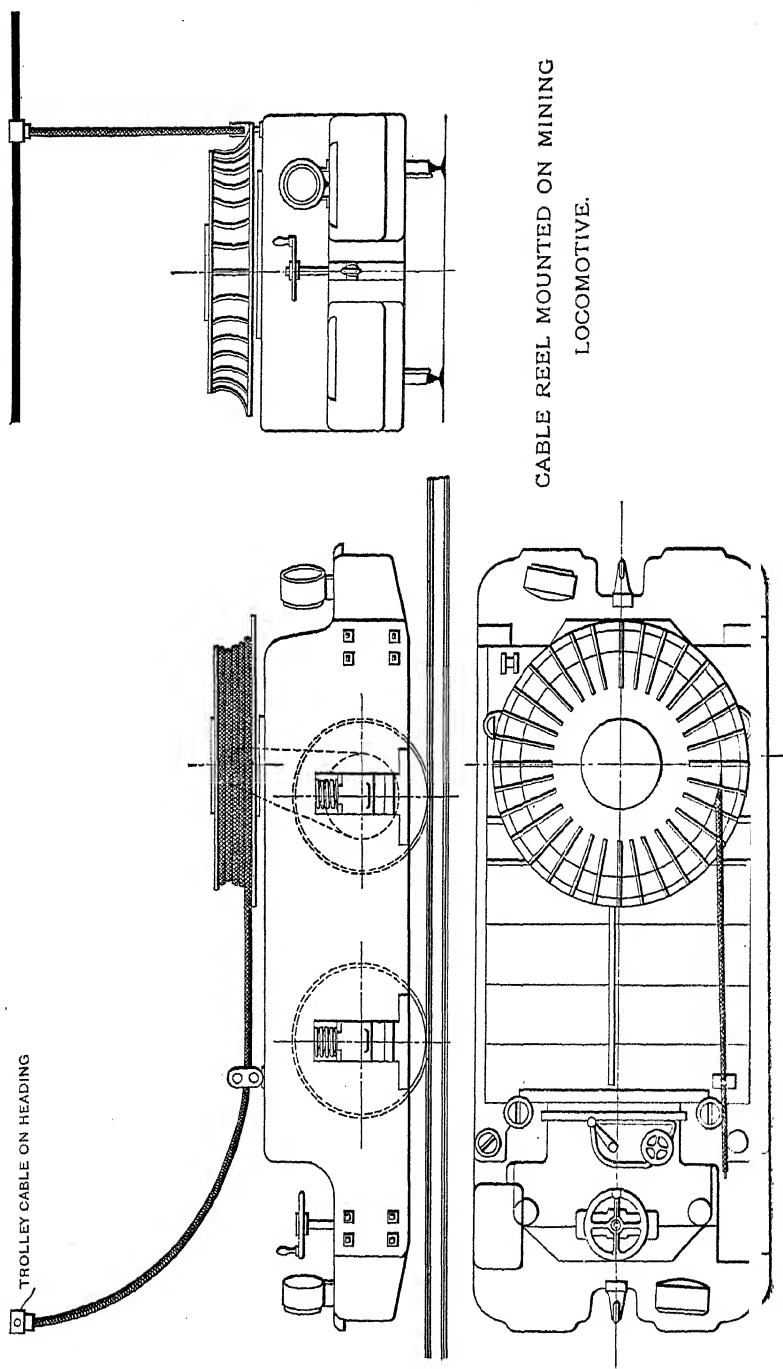
Electric locomotives, equipped with storage-batteries, have been tried for this class of service, and have also proved unsuccessful. At first thought, it might seem very simple to provide the locomotive with a storage-battery of such a voltage that the batteries would receive their charge whenever the locomotive operates from the trolley in the ordinary manner. Since the locomotive would operate from the trolley-wire on the main entries and branch headings during at least two-thirds of the time, it might be assumed that the batteries would always be fully charged. In practical operation, however, it is found that the voltage at the locomotive varies continually between very wide limits (on account of the line-drop); and when it falls below the potential of the batteries, they very promptly discharge into the line and assist the generating station. The practical result follows that it is impossible to keep the batteries charged in this manner, except by maintaining a perfectly uniform voltage throughout the entire system, which would entail an enormous and prohibitive investment for line-feeders. A complicated arrangement of voltmeters, ammeters and switches might be employed; but the fact remains that the motorman has but two hands, and they are engaged with the ordinary operating-device. Again, it would be possible to regularly charge the batteries from a separate plant, installed especially for that purpose; but, if the locomotive were in continual service (and that is presumably the purpose for which it is intended), the limitations of size and weight would neces-

sitate several sets of batteries. In either case, the batteries are expensive, and the cost of maintenance and depreciation for this severe service would be from 60 to 100 per cent. per year. The batteries must necessarily be placed on the top of the locomotive, where they prevent ready access to the motors; and, in actual service, no little trouble has been experienced by the spilling of the acid solution. Furthermore, the fumes from the cells render the occupation of the operator disagreeable in the extreme.

To meet the demand for a gathering-locomotive, the engineers of the General Electric Co. have designed a "cable-reel device" (see Fig. 1), which may be applied to standard electric mine-locomotives. This device is, briefly, a vertical reel of relatively large diameter, containing a flexible insulated cable. The end of the cable is connected to the trolley-circuit and the current is conducted to the controller on the locomotive through a suitable contact at the center of the reel. Iron rails are now quite generally used for room-haulage, in place of wooden rails; and a single-conductor cable with rail-return would ordinarily be employed.

The reel is driven by means of a sprocket-chain and bevel-gearing, connected with one of the axles of the locomotive. The gearing consists of a small pinion on an intermediate shaft and a large horizontal gear. The reel proper is not rigidly attached to the horizontal gear, but is driven therefrom by the friction due to its weight. The horizontal intermediate shaft, driven by the sprocket-chain from the main axle, is in motion whenever the locomotive is running, and carries two miter-pinions which are stationary, except when the reel is in use. Each of these pinions, within which the intermediate shaft ordinarily revolves freely, is provided with a clutch, operated by a lever located conveniently near the motorman. These clutches are automatic to the extent that they will positively operate the pinions in but one direction. At the entrance to a room, for instance, the trolley-pole is fastened down; the cable is connected to the trolley-circuit, and one of the clutches is thrown in by moving the lever to one side from the central "off" position. As the locomotive proceeds towards the face of the room, the cable unwinds from the reel, and the friction of the

FIG. 1.



various parts is sufficient to preserve a tension on the cable of about 30 lbs., which effectually prevents "kinking."

The instant the locomotive starts in the opposite direction (toward the heading), the pinion is positively driven from the intermediate shaft by means of the automatic clutch. A motion is consequently transmitted to the reel and the process of rewinding the cable begins. The direction of the locomotive is reversed in the room without any manipulation of the clutch-lever, which was adjusted before leaving the heading. If the room branches off from the heading in the opposite direction, the other pinion and its clutch are employed;—the two pinions being designed for opposite directions of positive rotation. In other words, the operations of reeling and unreeling the cable are independent of the direction in which the locomotive is moving.

As stated before, a tension of approximately 30 lbs. is maintained on the cable while unreeling. *The essential feature* of this device is the method by which the same result is accomplished while reeling on the cable. If the reel were positively driven, it would not wind the cable on with sufficient rapidity at first, and the locomotive would be liable to run over and cut the cable. Assuming that this did not occur, as the diameter of the reel increases by the additional layers of the cable, the speed at which the cable is taken on would be too high, and it would be broken. As already stated, the connection between the reel proper and the driving-gear is due entirely to the friction between large annular surfaces. The ratio of the driving-gearing is such that, except for the friction-surfaces, the peripheral speed of the reel would be about 25 per cent. higher than the linear speed of the locomotive. It is obvious, therefore, that when the cable is being wound on to the reel there is always a slipping between the friction-surfaces, with the result that the same tension of 30 lbs. is maintained on the cable at all times. This is the essential feature of the whole mechanism, since it is practically impossible to run over or part the cable, and there is always sufficient tension to prevent "kinking." The automatic pinions are also of prime importance, for by means of these no manual adjustment of the clutches is necessary while the locomotive is operating in a room. The

impossibility of damaging the cable was demonstrated in experimental tests by reversing the locomotive repeatedly at full speed.

Another important development, to which attention is invited, is in connection with the generation and distribution of power for the operation of electric coal-mining machinery. There is a striking analogy between an electric mine-haulage plant and a street-railway system; in fact, some of the principal mine-haulage plants do not suffer, by comparison in size, with street-railway systems of considerable magnitude. This statement is forcibly illustrated by the following data regarding the mines of the Berwind-White Coal-Mining Company at Windber, Pa., all located within a radius of about three miles:

Generating capacity in eight power-houses,	2900 kw.
(or nearly 4000 H. P.)	
Number of miles of underground trolley-lines,	75
Number of mine-locomotives,	44
Aggregate H. P. of mine-locomotives,	4400

For several years it has been customary to operate electric railways from central stations generating high-potential, alternating currents, which are transmitted to sub-stations and there converted into direct current at 500 volts. The mine-haulage plant installed at Ehrenfeld, Pa., by the Webster Coal & Coke Company, is, however, the first to embody the advantages of underground transmission with alternating currents at a high voltage. The haulage at this mine was accomplished by the rope-system and mules until the main heading attained a length of nearly two miles, when the necessity of substituting a more efficient and flexible system became very apparent. Since the main heading will eventually extend five miles underground, there was practically no alternative; and the electric-haulage system was adopted.

In accordance with the usual practice, ordinary direct-current generators would have been installed at the mouth of the mine. On account of the length of the main heading and the magnitude of the operation, however, this system would have entailed a prohibitive investment for feeder-copper, especially in view of the low operating-potential, namely, 250 volts. The system

recommended by Mr. L. B. Stillwell, the consulting engineer, and adopted by the officials of the Webster Coal & Coke Company, embodies the generation of a three-phase, 25-cycle, 5600-volt alternating current near the pit-mouth. This is transmitted to a sub-station in the mine near the center of actual operations, and 9000 feet from the power-house, where, by means of static transformers and a 250 kw. rotary converter, a direct current at 275 volts is delivered to the haulage-circuits.

The power-house contains two 150 kw. General Electric belt-driven, double-current generators, separately excited and delivering, from one side, direct current at 275 volts, and from the other, three-phase alternating currents at 165 volts. The ratio between the alternating and direct potential of a double-current generator is inherently fixed, and, in the case of a three-phase machine, is approximately 0.60. The direct current at 275 volts is fed into the outer end of the haulage-circuit in the usual manner; one terminal of the generator being connected to the trolley-circuit and the other to the rail-return. The potential of the alternating currents is raised from 165 to 5600 volts by "step-up" transformers. The transmission-line consists of a three-conductor, No. 6 B. & S., rubber-insulated, lead-covered cable, laid in the return air-way, which parallels the main heading.

The "step-up" transformers in the power-house and the "step-down" transformers in the sub-station are oil-cooled and are duplicates. Taps are brought out from the primary windings in order to compensate for the line-loss; and, consequently, the direct-current voltages of the rotary converter and the generators at the power-house are the same. The transformers are connected in "delta," so that the temporary disconnection of one of them cannot affect the system, beyond overloading the remaining two.

The double-current generators in the power-house are simply rotary converters, being duplicates of the converter in the sub-station, with their shafts extended to receive the driving-pulleys. It is proposed, as occasion may require, to remove these two generators from the power-house and establish additional rotary converter sub-stations in the mine beyond the



present sub-station. A 300 kw. engine-driven, revolving-field alternator has recently been installed in the power-house, and delivers 5600-volt three-phase currents direct to the transmission-line without the interposition of "step-up" transformers. A duplicate unit will be added at a later date, and direct current for the outer end of the haulage-circuit will probably be supplied by a rotary converter, located in the transformer-room of the power-house; although it may prove advisable to retain permanently for this purpose one of the double-current generators. By using the double-current generators a very considerable saving in the original investment was effected. These machines have operated in the most satisfactory manner, and their regulation with rapidly-varying alternating and direct-current loads has been surprisingly good.

In many mines the conditions are such that double-current generators would constitute an ideal permanent installation. The alternating-current induction-motor, with its "squirrel-cage" armature, is pre-eminently suitable for use in coal-mines, on account of the absence of the commutator and of armature-insulation. The Davis Coal & Coke Company, for instance, with extensive operations in West Virginia, use direct current for mine-haulage only. For all other purposes, alternating-current induction-motors are employed, and generators of both types are installed in each of the power-houses. By means of the double-current generator, both classes of service may be operated from one machine, while at the same time the advantages of high-voltage transmission may be secured by the use of "step-up" transformers. The writer ventures to predict an extensive field for the double-current generator in mining-work.

The principal advantage of the alternating-current system lies in the possibility of employing high voltages, since the cost of transmission copper varies inversely as the square of the voltage. The voltage of the direct-current system is practically limited to about 750, on account of commutation. The voltage of the alternating-current system, however, is limited only by the insulation of the transmission-line. In the open air, voltages as high as 50,000 are in successful every-day use, transmitting power for commercial purposes through distances as

great as 150 miles. In mining-work, however, involving underground transmission-lines, the limit may be placed at approximately 10,000 volts.

In the installation described above, a cable containing three No. 6 B. & S. wires transmits 350 H. P. with a line-loss of barely 5 per cent., and delivers direct current to the trolley circuit at 275 volts. On the other hand, if the direct-current system were used, with a voltage as high as 300 at the powerhouse and 250 at the center of distribution (*i.e.*, 16.6 per cent. line-loss), the transmission-line would consist of eighteen No. 0000 feeders on one side of the circuit and six No. 0000 feeders, in addition to the bonded double track, on the other side. With the alternating-current system, the copper for 5 per cent. loss weighs 2200 lbs.; while with direct-current system the copper for 16.6 per cent. loss would weigh 138,500 lbs., and for 5 per cent. loss approximately 560,000 lbs.

The system of power-distribution outlined above is especially suitable for the operation of several neighboring coal-mines from one central power-plant. The United States Coal & Coke Company are at present installing an interesting plant of this nature in the Tug River field, near Welch, W. Va. The initial installation consists of two 400-kw., 6600-volt, 25-cycle, 3-phase alternators, operating three 150-kw., 275-volt rotary converters, together with induction-motors for driving coal-crushers and conveyors. At first, three mines will be operated from this central power-plant; but other openings will be added from time to time. Mr. Stoek has already described a similar installation now being made by the Coal Department of the Delaware, Lackawanna & Western Railroad in the anthracite-field. The adaptation of the three-phase alternating-current system of power-distribution to coal-mining is especially opportune in this era of inception of large enterprises, and on account of the many consolidations of independent operations in the same district.

## Compressed-Air Motors for Gathering Cars in Coal-Mines.

BY BEVERLEY S. RANDOLPH, FROSTBURG, MD.

(Albany Meeting, February, 1903.)

WHILE our coal-mining practice, in regard to hauling on main roads, has advanced very rapidly in recent years by means of compressed air, electricity and ropes, that of gathering from rooms or working-places has remained almost stationary. Few large mines are without some method of mechanical haulage on main roads, but the gathering from working-places is still done almost entirely by means of animals or men.

An effort to improve this has been made recently by the Consolidation Coal Company, under the direction of the writer, at its mines in the Georges Creek region. The seam worked is the Pittsburg bed, known locally as the "Big Vein." It is from 8 to 12 feet thick. Immediately overlying the coal there is from 5 to 6 feet of "rashings," consisting of thin alternating beds of shale and coal, which disintegrate rapidly on exposure to the air, and making a very treacherous roof.

Rooms are driven from 12 to 15 ft. wide, with a single track close to one side. A line of posts is placed just far enough from this side to leave a clear space for the mine-car with a driver at the brake. This brings the post not far from the middle of the cross-bar, and provides more effectual support than if placed at the end. The tracks in these rooms are usually of 4 in. x 4 in. oak scantling, though of late years, owing to the advance in the price of lumber and the reduction in steel rails, more of the latter are being used. This track is laid by the miner. His pay for the work is included in the price per ton for mining. It is therefore, as a rule, unskillfully and often carelessly laid, and cannot be relied upon to carry safely any weight materially greater than the loaded mine-car.

Compressed air was already in use on the main roads, driving motors weighing 30,000 pounds each, which were used to bring the mine-cars that had been assembled from the rooms

to the slopes, through which they were hauled to daylight by means of ropes.

At the suggestion of the writer, the Baldwin Locomotive Works designed a motor, having the dimensions and general character shown in accompanying drawing (Fig. 1), and guaranteed not to weigh more than 8000 pounds when charged. Loaded mine-cars frequently weigh 7000 pounds gross, and their outside dimensions are then practically the same as the motor just mentioned.

Five of these machines were placed in the Company's Ocean No. 3 mine (Hoffman), displacing a number of mules, but leaving 19 still working. This opportunity was embraced to make a close comparison between the two methods of gathering.

The mules working in the North Heading and the South Heading deliver their cars directly to the rope on the slope. The other mule-routes deliver to the heavy motors mentioned above, as do all the motor-routes. The mules used weigh from 1200 to 1400 pounds, and are the best obtainable. Mine-cars weigh 1600 pounds, and carry an average of 2.4 long tons.

The following table shows the work performed by the mules during a period of 18½ working days in the month of December, 1902:

Route.	Cars Moved.	Average Haul.	Constant.	Tons Moved 1000 ft.
South heading.....	1119	2900 ft.	$1\frac{24}{1000}$	7788.24
North " .....	268	1300 "	"	886.16
1st Cross.....	1629	2100 "	"	8210.16
2d Cross.....	3042	1100 "	"	8030.88
3d Cross.....	747	400 "	"	717.12
Total.....				25,582.56

This represents a total of 339 days' work for one mule.

The company's accounts show a cost of \$1.15 per day for each day worked by a mule, including expense of replacing worn-out animals. Drivers are paid \$1.98, and there is one with each mule. This makes a cost of \$3.13 per day for each day worked by a mule. The cost per ton hauled 1000 feet would therefore be

$$\frac{339 \times \$3.13}{25,582.56} = 4.15 \text{ cents.}$$



For the work of the motors during the same time we have :

Route.	Cars Moved.	Average Haul.	Constant.	Tons Moved 1000 ft.
Tippens.....	1122	2300 ft.	$\frac{2.4}{1000}$	6193.44
Scobies.....	1073	2050 "	"	5279.16
1st Klondyke.....	1147	1835 "	"	5046.80
2d ".....	1032	1800 "	"	4458.24
3d ".....	1147	1865 "	"	5138.56
4th ".....	114	1992 "	"	544.92
Total.....				26,661.12

This work was done by the five small motors operated by compressed air, working a total of 94 days.

This plant is supplied with steam by a battery of boilers, which also supplies steam to the large pumps. The plant consists of the following items, with their approximate first cost:

One straight-line Norwalk air-compressor, 18 and 28 compound steam, 18½, 13½ and 6½ three-stage air, 30-in. stroke, . . . . .	\$5,300
5600 feet of 5-inch pipe, . . . . .	5,600
3100 " 2½-inch " . . . . .	1,700
1000 " 1½-inch " . . . . .	300
2 motors, 30,000 lbs. each, . . . . .	6,000
5 " 8,000 lbs. each, . . . . .	10,000
Estimated proportion of boilers, . . . . .	1,000
Installation, . . . . .	4,000
	<u>\$33,900</u>

Allowing \$3000 per year for interest and depreciation, to be earned in 300 working-days, would justify a charge of \$10 per day from this source against the entire plant.

This same compressor also drives the large motors mentioned above, which weigh 30,000 lbs. each (60,000 lbs. for the two); the five small machines weigh 8000 lbs. each (40,000 lbs. for the five). Dividing the general expenses according to the weight would result in four-tenths being charged against the small motors.

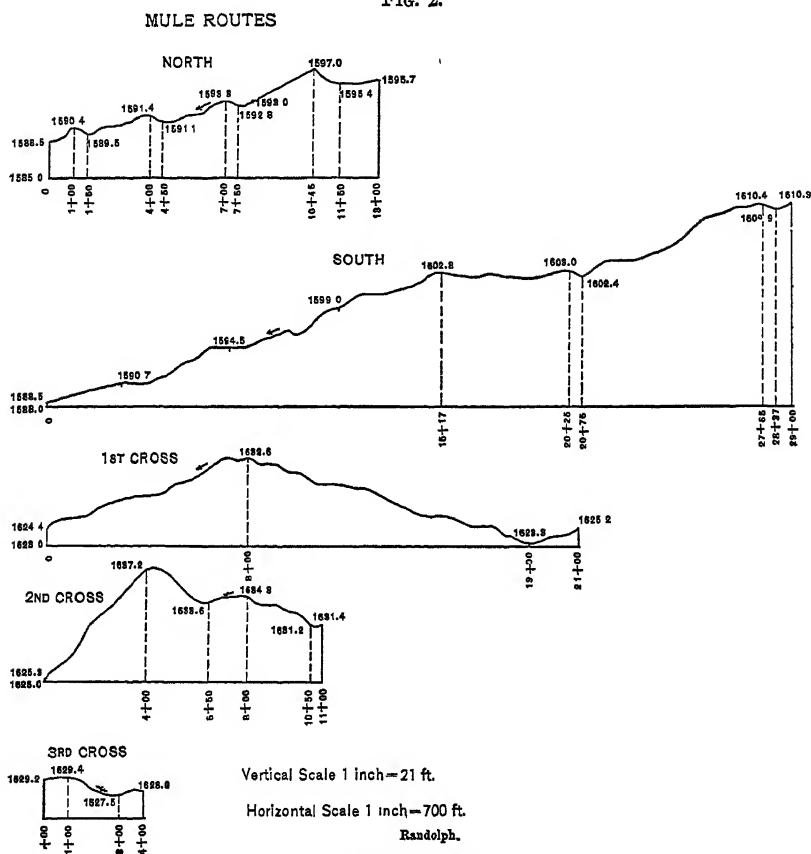
These general expenses may be summed up as follows per day :

Coal, 4 tons @ \$1, . . . . .	\$4.00
Fireman, . . . . .	2.00
Mechanic in charge of compressor, . . . . .	2.50
Interest and depreciation, . . . . .	10.00
	<u>\$18.50</u>

The daily cost of operation of the five small motors would then be

5 Motormen at \$2.67, . . . . .	\$13.35
5 Brakemen at \$2.03, . . . . .	10.15
General expenses, \$18.50 x .4, . . . . .	7.40
Repairs and oil, . . . . .	3.00
	<hr/>
	\$33.90

FIG. 2.



Mule Routes.

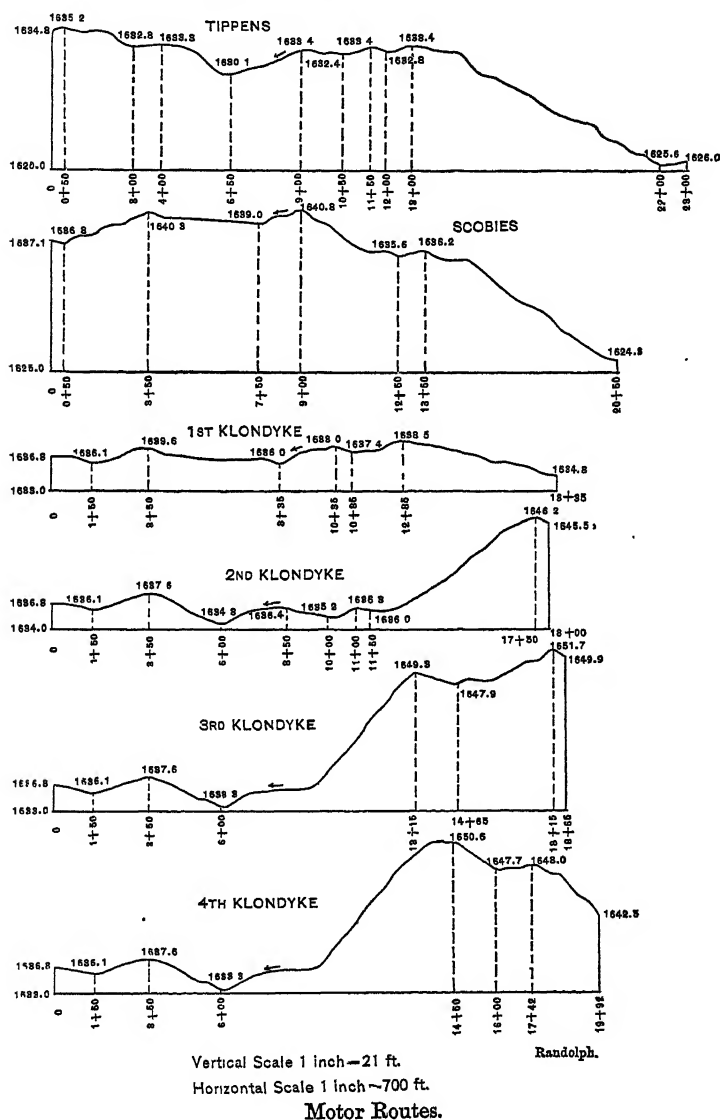
Dividing this among the five machines would give \$6.78 per day for each machine, and the cost per ton moved 1000 feet would be

$$\frac{6.78 \times 94}{26,661.12} = 2.44 \text{ cents.}$$

In the matter of continuity of service, the motors show to great advantage. A broken-down motor can usually be re-

paired over night, while an injured mule can only be replaced by a new one that must usually be broken and inured to the

FIG. 3.  
MOTOR ROUTES



work before he is thoroughly efficient, entailing loss of time and output in each case.

In the actual placing of cars in the workings, the motor has



little or no advantage over the mule. After the train is made up, its higher speed and larger load place the mule at a great disadvantage.

The showing of the motors may, therefore, be expected to be better with long hauls than with short.

Owing to the fact that it is the practice in this mine to send the same motor or mule on different routes, depending on where miners may be loading coal, it is impracticable to present any discussion of this feature from the data at hand.

Figs. 2 and 3 show the profiles of the various routes. The loads move from right to left. From this it will be seen that, so far as grades are concerned, the motors are at a decided disadvantage in this comparison. The elevations refer to mean tide at Baltimore.

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### Note on the Influence of the Rate of Cooling on the Structure of Steel.

BY ALBERT SAUVEUR AND H. C. BOYNTON, CAMBRIDGE, MASS.

(Albany Meeting, February, 1908.)

IN the course of some experiments conducted in the Metallographical Laboratory of Harvard University, some interesting facts were brought to light which appear to be worth recording in advance of a more elaborate and exhaustive paper which the authors hope to present to the Institute in the near future.

Fig. 1 shows, under a magnification of 100 diameters, the microstructure of the cross-section of a steel bar  $\frac{1}{2}$ -inch square, containing 0.52 per cent. carbon, heated to 1100 degrees C. and cooled slowly with the furnace. Fig. 2 illustrates the structure of the same steel, heated to the same temperature but cooled more rapidly in the air.

Both samples were cut from the same bar and heated side by side in the furnace. When a temperature of 1100 degrees C. had been reached, one sample was taken out of the furnace and allowed to cool in the air, while the other was cooled with the furnace. The only difference, therefore, in the treatment of both samples will be found in their respective rate of cooling. This variation in the rate of cooling resulted, as shown

here, in a very marked difference in the structure of the two samples.

The air-cooled sample has assumed a sharply defined network structure, while the furnace-cooled specimen possesses a structure which, for lack of a better term, we shall call a granulated structure.

Similar experiments were repeated a great many times with identical results. Air-cooling from temperatures exceeding 1000 degrees C. always resulted in the formation of a network structure, while very slow cooling in the furnace, from similar temperatures, was always accompanied by the formation of a granulated structure.

The examination of these two structures cannot fail to impress us with the important part played by the rate of cooling from a high temperature upon the constitution, and, therefore, upon the properties of medium hard steel, and probably of all steels.

More thoughtful consideration will lead to other conclusions and speculations which are not without scientific as well as practical interest.

It is a well-established fact that all the carbon present in undersaturated or hypo-eutectic steel is included in the constituent pearlite. It is also generally stated that pearlite has a constant composition, its carbon content being in the neighborhood of 0.8 per cent. Finally, undersaturated steels, when unhardened, are always described as being made up of ferrite and pearlite, which implies the assumption that the dark constituent of these steels is always made up of pearlite. Upon examination of Figs. 1 and 2, however, it will be found that the dark constituent occupies a much larger proportion of the mass in the case of the air-cooled sample. A rough estimate will show that the dark constituent occupies a little more than one-half of the area of Fig. 1, while it occupies at least 90 per cent. of the area of Fig. 2. It is evident, therefore, that the carbon in Fig. 1 has segregated into a smaller bulk, or, in other words, the dark constituent of the furnace-cooled sample contains a larger proportion of carbon than the dark constituent of the air-cooled sample. If we call the dark constituent of both samples pearlite, then we must infer that the composition of pearlite, instead of being constant, varies with the rate of cool-

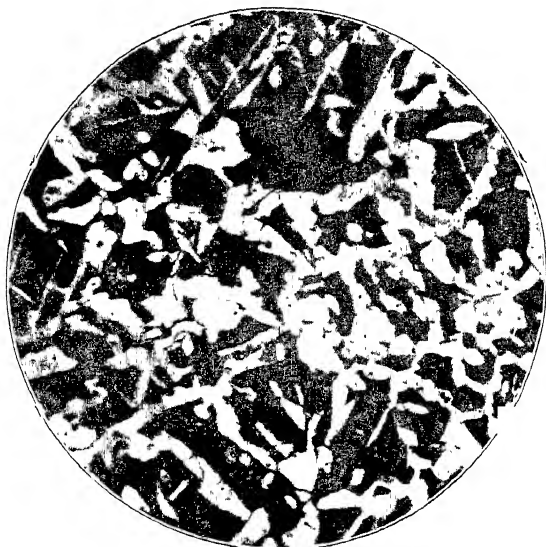
ing. Such conclusion, however, would be so much opposed to many important evidences which we have concerning the constitution of pearlite as to be quite untenable.

To throw more light upon the nature of the dark constituent of Figs. 1 and 2, the same samples were examined and photographed under higher magnifications, as shown in Figs. 3 and 4. The high-power photograph of the furnace-cooled sample includes a particle of ferrite and some of the surrounding dark constituent. It will be noticed that the dark constituent exhibits everywhere the characteristic structure of pearlite, which is also that of nearly all eutectic alloys: It is made up of parallel plates alternately of one and the other constituent (of ferrite and cementite in the case of steel). In Fig. 4 is shown a high-power photograph of one of the meshes of the network of the air-cooled sample. The dark constituent is only partially laminated; in many places its structure is ill-defined; it does not possess, to the same degree, the structural characteristics of eutectic alloys.

The inference from these considerations must necessarily be, that while the dark constituent of the furnace-cooled sample is true pearlite, the corresponding constituent in the air-cooled sample is not true pearlite; it contains more iron (more ferrite) than true pearlite, evidently because the time necessary for this excess of ferrite to segregate (to be expelled) was denied by the relatively rapid cooling of this sample.

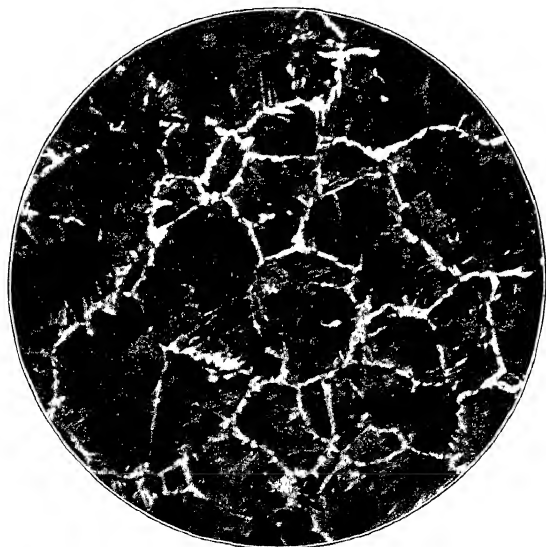
The mechanism controlling the formation of the structure is apparently as follows: At a high temperature, the whole of the carbon is diffused through, or dissolved in, the iron. Steel at this temperature has rightly been compared to a solid solution of iron and carbon. This solid solution can be retained in the cold (partially, at least) by sudden cooling, the resulting constituent being then called martensite. Upon slow cooling, however, the solution is unable to retain the whole of the iron; as the metal cools, some ferrite falls out of solution, and if the cooling be sufficiently slow a stage of equilibrium is reached at each temperature. When the critical temperature is attained, the remaining solution of iron and carbon reaches the composition of the eutectic alloy (that is, of pearlite), and in passing through the critical range the carbon enters into combination with some of the iron still holding it in solution, to form the

FIG. 1.



Heated to  $1100^{\circ}$  C. and cooled with furnace (100 diameters).

FIG. 2.



Heated to  $1100^{\circ}$  C. and cooled in air (100 diameters).

FIG. 3.



Same sample as Fig. 1 (1000 diameters).

FIG. 4.



Same sample as Fig. 2 (1000 diameters).

carbide  $\text{Fe}_3\text{C}$ , or cementite. If the cooling be not sufficiently slow, however, to allow this stage of equilibrium to be reached, some of the iron is retained by the dark constituent, which therefore never reaches the composition of true pearlite, and which might then be called dilute pearlite or ferrous pearlite.

It is evident, from what precedes, that in order to induce the formation of true pearlite in steel, the metal must be cooled sufficiently slowly from a temperature exceeding say 1000 degrees C., to, and past, the critical point; the rate of cooling below the critical point being quite immaterial. If the time necessary for the formation of true pearlite be denied, then the dark constituent present in steel is of a different character: it contains a smaller proportion of carbon, and assumes, imperfectly and locally only, the structural characteristics of pearlite.

It should also be remembered that since the formation of true pearlite depends upon a sufficiently slow rate of cooling, this constituent will be more readily formed in large pieces than in those of small dimensions.

The authors are well aware that other writers have alluded to this character of the dark constituent of relatively quickly cooled steel, but they trust that they have brought these facts to the attention of those interested in steel more forcibly than has ever been done before.

The dilute or ferrous pearlite just alluded to, resulting from the relatively quick cooling of steel from a high temperature, recalls the constituent which Osmond has called sorbite, the existence of which has been and is still doubted by some metallographists. Sorbite, however, results essentially from a relatively quick cooling *while the metal is passing through the critical range*, in such a way that the critical transformation which results in the production of true pearlite is not fully completed. The constituent being considered here, on the other hand, is the result of relatively quick cooling *above the critical point*, so that it retains more ferrite than true pearlite, and occupies a larger proportion of the mass. Osmond's sorbite does not necessarily contain more ferrite than true pearlite. Looking into the matter more closely, however, it will be seen that a relatively slow cooling to the critical point almost necessarily means the same rate of cooling through the critical point; so

that the characteristics of sorbite will also be developed in this dilute pearlite, unless the cooling be conducted in a very peculiar way, consisting of rapid cooling to the critical point, and of slow cooling through that point. It appears to the writers that the dilute pearlite with which they are now concerned may properly be called sorbite. To select another name for it might lead to confusion.

Most forged steel implements are air-cooled, which means, in the majority of cases, too rapid cooling for the production of true pearlite. The dark constituent of forged steel which is always referred to as pearlite is seldom true pearlite; in the majority of cases it is sorbite. Witness the network structure so frequently found in steel rails containing about 0.50 per cent. carbon, and which is so similar to the structure shown in Fig. 2.

It was long ago proposed to divide pearlite into two varieties, the lamellar and the granular. It appears, however, that if the dark constituent does not exhibit the laminated structure of pearlite, it consists of sorbite, and not of true pearlite. Granular pearlite is not pearlite at all, but sorbite.

It is evident, moreover, that the carbon content of sorbite will vary with the rate of cooling. It is, as Osmond has rightly said, a transition constituent, marking a step in the transformation of martensite into pearlite. The quicker the cooling, the more it will approach the nature of martensite, and *vice versa*.

Several interesting questions are suggested by this varying composition of the dark constituent of unhardened steel. Some of them will be briefly mentioned here.

1. What is the difference between the properties of a steel whose dark constituent consists of true pearlite, and those of the same metal whose dark constituent is made up of sorbite?

Samples of steel containing 0.55 per cent. of carbon, and to which the two kinds of structure had been imparted, were tested, and it was found that the network-structure—that is, the structure in which the dark constituent consists of sorbite—corresponded to a greater elastic limit, and greater tenacity, but to a smaller ductility. Some of the results are given below :

Treatment of Sample.	Character of Dark Constituent.	Elastic Limit. Lbs. Per Sq. In.	Tensile Strength. Lbs. Per Sq. In.	Elongation. Per Cent. in 8 Inches.	Reduction of Area. Per Cent.
Heated to 1150° C. and cooled with furnace.....	Pearlite.	39,901	81,162	17	27.61
Heated to 1150° C. and cooled in air.....	Sorbite.	53,260	99,979	12	20.55

2. What are the properties of pearlite compared to those of sorbite? To answer this question 2 samples of steel, in the shape of bars  $\frac{1}{2}$ -inch square, should be procured, containing respectively about 0.60 and 0.80 per cent. of carbon, and as far as possible containing the same percentages of other impurities. The first sample should be heated to say 1000 degrees, and cooled in the air. We should expect to find that it consists then of a mass of sorbite, as time had been denied for the excess of iron to segregate as ferrite. The second sample should be heated to the same temperature but cooled slowly in the furnace. It will then consist of a mass of true pearlite.

The physical testing of these two samples will indicate the respective properties of pearlite and sorbite (at least of the sorbite formed under the conditions just indicated).

3. Seeing that sorbite is a transition form between martensite and pearlite, does it not also contain its carbon in a transition form between hardening carbon (the condition of carbon in martensite) and cement carbon (the condition of carbon in pearlite), or partly as hardening and partly as combined carbon? Careful analysis by the colorimetric method of samples of the same steel, but containing respectively sorbite and pearlite, should throw much light upon this point. It can be reasonably expected that the analysis of the pearlite steel will give higher results than those of the sorbite steel.

4. When it is attempted to judge of the amount of carbon present in a sample of steel by the appearance of its microstructure, the variation of the area occupied by the dark constituent, with the rate of cooling, as described in these pages, should be carefully borne in mind, otherwise serious errors will be made. It would appear that in order to apply this method



with any degree of accuracy, it is necessary first to treat the samples so as to cause the formation of true pearlite. With this precaution, the authors believe that the method will give results comparing favorably in accuracy with the colorimetric methods.

These points and a number of others are being investigated in the Metallographical Laboratory of Harvard University, and the authors hope to be able, before long, to publish some of their results.

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### Notes on Accidents Due to Combustion within Air-Compressors.

BY ALBERT R. LEDOUX, NEW YORK CITY.

(Albany Meeting, February, 1908.)

WITH the improvements in design and efficiency of machinery the element of danger in its use is becoming less, but it is a question whether the strain involved in operating modern plants is not increasing somewhat the danger of accident due to human fallibility.

I propose to describe a somewhat uncommon casualty resulting from a not infrequent incident in the use of air-compressors. It is not necessary to state the location of the mine in which the accident occurred nor the makers of the compressor, and the object of this brief paper is more to draw out suggestions from members more familiar than am I with the practical handling of mining machinery, than to point a moral.

The compressor in question was a three-drill machine of standard make. At the time of the accident it was furnishing air to a single drill working in an upraise from a well-ventilated tunnel, and giving ventilation to a winze below the tunnel where two men were hand-drilling. The compressor also furnished power to a small hoist in the winze, but the hoist was only operated occasionally, and never while the drills were working. The drills were located about 1200 ft. from the compressor. The engineer testified that he never had been short of air, and that there had been no complaints that the

machine was inefficient. The engineer testified, further, that the water used in cooling the air-cylinder had never become greatly heated, but that he used a mixture of good cylinder-oil and of a lighter grade known as "Atlantic Red." The valve of the compressor was set to blow off at 80 lbs. He was eating his luncheon in the boiler-room when he heard a "crack like a pistol," and, going into the engine-room, found water spurting out of the jacket about two feet from the end of the compressor; he tightened the jacket and stopped the leak, and found that the jacket was perfectly cool. He next noticed that "grease began to fry on the pipe and receiver."

He next saw that the air-pipe had become red-hot, the heat extending to a point where the pipe went through a wooden partition, setting fire to it. Then he noticed that the pressure was going down "just as quick as if someone had opened a valve outside," which, in fact, is what happened. He stopped the engine, but, getting the signal for more air, started up again. It is well to note at this point that the intake of the compressor was in the engine-room, the temperature of which usually stood at 115° F.

Let us now see what occurred in the mine. What happened at the bottom of the winze cannot be better told than in the testimony of one of the miners at the Coroner's inquest: "I was turning a hole and my partner was striking the drill; he says, 'We ain't got much air here this morning; I will ring for some more air.' He rang, started to strike again, and struck two or three blows; straightened up and took a couple of breaths of air, and started in to strike again, and then quit. I had been joshing him, and says, 'You ain't as tough as you used to be.' I stood up, picked up a pick, struck two or three blows, and felt that the air was bad. Just then the air stopped, and the hoist-tender hollered to us that there was no air and we had better come up. I looked around and seen my partner standing in the corner. He was all trembling, and I caught him as he fell over."

These two men were gotten up by heroic work on the part of comrades, and their lives were saved. But not so fortunate were the men in the upraise. There were four of them here, and when they felt the air getting bad they opened the valve full,—

of course, only increasing the difficulty. They were experienced miners, and at first could not understand what was the matter with them, because their candles continued to burn as usual. This was due, undoubtedly, to the fact that they were working in an upraise, and that the heavy carbonic-acid gas sank, and perhaps to the fact that carbonic oxide may have been generated through incomplete combustion or the reduction of the carbonic acid, first formed, by the glowing carbon in the pipe.

Two men were killed and four others barely escaped with their lives, as a result of the combustion of the oil, deposited carbon and organic dust accumulated in the compressor, receiver and pipe. Explosions of air-compressors due to this cause have been frequent, and lives have been lost thereby; and what is known as the "flaming" of compressors or cylinders is an every-day experience, and in some cases the rupture of the air-pipes, but, so far as I can ascertain, without the serious consequences described in the present case.

I shall leave to those better qualified than I the discussion of the best means of preventing such catastrophes. Among those which suggest themselves are the taking in of the air from a point where its temperature is as low as possible, the introduction of auxiliary coolers, the use of as heavy oil as possible,—yet never in excess,—the cleaning out of cylinders, receivers and pipes, and especially a warning to the engineer to be very sure, when he receives a signal for more air, that the actual shutting-down of the compressor may not be more essential. In this case, had he shut down it is probable that no lives would have been lost, for with the stopping of the air the miners would have at once returned to the tunnel level.

## Origin of Pebble-Covered Plains in Desert Regions.

BY WILLIAM P. BLAKE, TUCSON, ARIZONA.

(Albany Meeting, February, 1903.)

THE attention of travellers, upon the desert bordering the Great Colorado of the West, is often arrested by broad stretches of pebble-covered plains, or *mesas*, glittering in the sunlight from the myriads of polished surfaces, giving, at a distance, the appearance of a sheet of water. It is not alone the well-rounded, polished surface of these pebbles which commands attention, but, in addition, their nearly black or dark-brown color; and, above all, their uniform distribution in a level sheet, covering the plain in a continuous layer or pavement like a vast mosaic without sand or soil. Hundreds of square miles along the lower Colorado, especially in Yuma county, Arizona, and on the borders of the Colorado Desert in California, are thus covered. It is evident that the former extent of such pebbly plains was much greater than now, for the continuity of the *mesa* is broken into by the numerous dry *arroyos* or "washes" formed during exceptional showers or deluging rains.

In some places there is an underlying bed of pebbly conglomerate, a mixture of pebbles and sand; in others the pebbles rest on a sandy, earthy foundation. The rounded pebbles are distinctly alluvial in origin, and pertain to the ancient Colorado drainage system; but it is not comprehensible that such regular layers of polished pebbles could have been left by subsiding floods or river action.

Attention was early directed by me to these remarkable plains, first examined in 1853, when exploring for the United States and seeking a practicable route for a railway to the Pacific coast.\* The cut and polished surface of the pebbles

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\* "Report of a Geological Reconnaissance in California," vol. v. *Pacific R. R. Reports*, pp. 108, 112, 117.

and of loose rocks on the desert was then correctly explained as due to the attrition by wind-driven sand and dust, but an explanation of the pavement-like surface and the accumulation of pebbles and rock fragments in close contiguity was not attempted.

The object of this paper is to offer an explanation and to add to our knowledge of the phenomena of æolian denudation.

It appears certain that the surface-sheet of pebbles, and in many localities of small fragments of rock and bits of fossil silicified wood, is the result of an accumulation by the gradual removal, by the wind, of sand, silt or finer materials from around and below the pebbles, undermining them, removing their support, and permitting them to sink to a lower level, where they accumulate by concentration until the whole surface is closely covered by fragments too large and heavy to be moved away by the wind. Such a deposit of pebbles or fragments may thus represent all that were originally distributed through several feet of thickness of sand and lighter alluvions. In this way there may be a removal of a large amount of fine materials from the general surface of a region, lowering its level, until, by the complete covering and protection of the surface by the concentration of the heavier fragments, æolian denudation is arrested, and the further lowering of the surface is stopped.

## Zinc- and Lead-Deposits of Northern Arkansas.\*

BY GEORGE I. ADAMS, WASHINGTON, D. C.

(Albany Meeting, February, 1903.)

A PARTY, consisting of George I. Adams, of the United States Geological Survey, A. H. Purdue, of the University of Arkansas, and Ernest F. Burchard, was engaged, during the summer of 1902, in the study of the zinc- and lead-deposits of Northern Arkansas. An extensive report on this field is now in preparation. The following preliminary statement of results and conclusions is made in advance of the final publication, which will be accompanied by detailed geologic maps and other illustrations and a fuller treatment of the subject.

*Location.*—The Northern Arkansas field embraces Marion county and adjacent portions of Boone, Baxter, Newton and Searcy counties. Outside of this area there are a few scattered mines, notably in Sharp and Lawrence counties. The mining development lies north of the Boston mountains, in what is known as the Ozark Plateau. The country has a broken surface, as a result of dissection by streams, and there are numerous exposures of the mineral-bearing horizons in the valley slopes. This has greatly facilitated prospecting.

*History.*—Lead-ore was discovered in Northern Arkansas by the early explorers and pioneers, and at first was utilized for rifle-bullets. Later, it attracted considerable attention, and in the '50's was smelted in the vicinity of Lead Hill. There was a revival of the industry during the '70's, but the cost of transportation was so great that it was practically abandoned.

Zinc-ores at first attracted little attention, probably because they were not so well understood. Before the Civil War, however, some zinc was smelted. In the '80's, prospecting was carried on generally throughout the field, and resulted in finding zinc at so many places and so readily accessible that

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\* Published by permission of the Director of the U. S. Geological Survey.  
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about 1899 there was what might be called a "rush" into the field.

*Production.*—From the information contained in published reports, and gathered by inquiry, it has been estimated that the output of the Northern Arkansas district up to and including the year 1900 was 1500 tons of zinc-ore and 500 tons of lead. In 1901 about 500 tons of zinc were marketed, and in 1902 about 1000 tons, or double the amount of the previous year. The production of lead during 1901 and 1902 was unimportant. There is, at present, considerable ore stored in the bins awaiting transportation facilities, and the production of the district promises to increase during the coming year.

*Present Development.*—The condition of the mining industry in Northern Arkansas has been largely governed by transportation facilities. Until recently no railroad had entered the field. In 1901 the St. Louis and Northern Arkansas railroad was built to Harrison, and since that time has been extended to the Buffalo river. Its proposed extension is by way of Marshall southeastward, in the direction of Little Rock. The Missouri Pacific is now building from Batesville along the valley of White river to Buffalo City, and is projected northwestward, by way of Yellville, into Missouri. The completion of the railroads will afford facilities for shipping, so that mines in which operations have been suspended will resume, and others which have attempted no development beyond prospecting are already erecting mills or determining more definitely the character of the ground preparatory to doing so.

*Prospective Development.*—That which has been done thus far, in the way of mining, is not a satisfactory test of the field. Some companies which have been organized have been promoted by men inexperienced in the production of zinc and lead. The expenditure of money in erecting mills and bringing in machinery, and the failure to market ore at a profit, because of the long wagon-hauls, has usually resulted in the suspension of operations. The ore-deposits of Northern Arkansas are not such as mining men generally are familiar with, and some have been misled by the results of the prospecting. The mixed character of the ores found in surface-workings has made it difficult to produce clean concentrates. With the continuance of deeper workings and the following of

the ore-bodies into the hillsides, the sulphides are found to predominate, and this difficulty largely disappears. While it is impossible to predict with certainty the future of the field, there are mines now opened which are capable of a large output, and many of the prospects are promising and well warrant fuller exploitation. With the completion of the railroads the Northern Arkansas district promises to assume its true commercial importance.

#### GEOLOGIC RELATIONS.

The ores are found in two formations. The lower of these is the Ordovician dolomites, and the upper the Mississippian limestones. The Ordovician rocks occur extensively in Baxter county, in the northeastern part of Marion county, and in the other portions of the field, along the valleys of the streams, where erosion has cut down to them. The Mississippian limestones lie to the south and southwest, forming an irregularly fringed and dissected belt, lying somewhat higher and extending to the base of the Boston escarpment. The Mississippian limestones formerly extended farther to the north and northeast, and overlaid the Ordovician, but have been removed to their present boundaries by the wearing away of the land-surface through the action of atmospheric agencies. In addition to these formations, which are the principal ore-bearing rocks, there are some thin formations found between the two in certain parts of the field; but for the immediate discussion of the problems connected with the ore-deposits they need not be described. The saccharoidal sandstone, however, which lies above the Ordovician dolomites, and accordingly separates the lower from the upper ore-bearing rocks, should be mentioned, since it is a convenient datum in this field. It is known locally as the sand-ledge, and is referred to in determining the horizons of the mines and prospects which occur below it in the Ordovician. To the southward, lying upon the Mississippian limestones, and accordingly higher in the geological column, are the shales and sandstones which are extensively developed in the Boston mountains.

#### *Structure of the Rocks.*

An examination of the mines of the district shows that the ore-bodies are related to two classes of structure, viz., simple



fractures and breccias. The rocks, considered broadly, are found to be nearly horizontal. They exhibit locally, however, undulating structure and occasionally well-marked dips. There are well-defined normal faults, which are later than the fracturing and brecciation above mentioned, but, excepting in certain instances, where fault-breccias have been developed, there is only a minor amount of mineralization along the fault-planes or in the material filling the normal fault-fissures.

The fracturing of the Ordovician rocks was produced by compressive forces, and in certain zones has a considerable vertical extent. A second and equally important result was brecciation, which was produced by the differential movement of the strata. The variation in the structure of the dolomite series, which is in places massively bedded and in other places thinly bedded, laminated, and even shaly, resulted in the lateral movement being taken up in varying degree by the individual beds. The motion was such as is produced by forces acting in couples. The brecciation is due to the tendency of the pieces, resulting from the breaking of certain brittle strata, to sheer past each other, or to rotate with the horizontal movements of the adjacent beds, so that the fragments are relatively displaced.

In the Mississippian limestones the compression produced, principally, fracturing and fissuring. The walls of the fissures not infrequently exhibit slickensides, which have been produced by the rocks moving past each other horizontally. The Mississippian limestones do not exhibit brecciation, excepting in fracture-zones, or where they have been crushed by the dragging of the beds along normal faults, which, in most cases, are due to a later adjustment of the rocks of the area.

*Origin of the Structure.*—The fractures and brecciation above mentioned are probably due to stresses induced at the time of the folding in the Ouachita\* mountains and Arkansas valley regions. At the close of the Carboniferous period the thick mass of sediments which had accumulated in what is now central Arkansas and eastern Indian Territory was folded in a manner which suggests that they were thrust to the northward. In the Ouachita mountains there is close-folding and thrust-faulting, while in the Arkansas valley region there are open folds. In the southern border of the Ozark region, and

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\* Or Washita.

particularly the area here under discussion, the generally horizontal position of the rocks was retained, but there was considerable movement of individual beds. This movement was one of accommodation, and resulted in fracturing, without marked displacement. It took place largely along the bedding-planes, and resulted in brecciation of the beds. The normal faulting in this area is of later date, and is probably due to the readjustment following the crushing, or to subsequent oscillations of level.

*Geologic Conditions Influencing the Circulation of Ground-water.*

The rocks which constitute the Ordovician series and the Mississippian limestones of the Northern Arkansas district may be considered as relatively quite permeable. There are local beds of shale in the Ordovician, through which water would not readily pass, and the Devonian, which has a very limited extent in the southern part of the field, is of about the same character and importance in controlling the path of the ground-water. They do not have a wide influence, since they do not form persistent horizons. Where they occur the solutions were, probably, diverted laterally, but no localization of ore-deposits seems to be directly due to them.

The shales lying above the Mississippian limestone, on the contrary, have sufficient thickness to make them an important factor in determining the movement of ground-water. Formerly, they extended from their present boundary near the base of the Boston mountains northward into Missouri, and covered a considerable part of the Ozark regions. Before they were removed they acted as a confining or limiting horizon. Water entering the Ordovician dolomites and Mississippian limestones, where they outcrop beneath the shales, and moving with the dip, as it passed southward, was under a hydrostatic pressure. There may have been a first concentration of the ores, due to this circulation; and, if so, it could have taken place from the upper portion of the Mississippian limestones to the bottom of the Ordovician. This reasoning may be appealed to in accounting for the ore-bodies now found in the Mississippian limestones near the base of the Boston mountains, where these rocks have recently been uncovered by erosion. It is possible, however, that the concentration could have occurred through

the agency of lateral circulation adjacent to the fractures in which the ore-bodies are found, without appealing to causes which are of such wide influence.

As erosion progressed, the shales and other formations lying above the Mississippian limestones were speedily removed from the more central portion of the Ozark region, so that the conditions which at first existed, as above outlined, were not long maintained. The main streams of the region, such as White river and its tributaries, soon cut through these formations, and the water, which may have formerly been under hydrostatic pressure, found issuance in their valleys. At the present time there are no upper confining shales in the northern part of the field, and this condition has prevailed for a long period. The surface-water has been free to descend into the Mississippian limestones and Ordovician rocks, or through the Mississippian limestones into the Ordovician, and the point of issuance of such portions as have reappeared in the surface-flow has been in the valleys of the larger streams. There, no doubt, has been lateral movement along bedding planes, through the more permeable strata, open and brecciated beds and with the surfaces of local shale-beds. The saccharoidal sandstone, which is a conspicuous formation and one which is relatively porous, has probably been a horizon of lateral movement, assisting in the transfer of the ground-water to places where it could find its way into the adjacent beds.

#### *Relations of Belt of Weathering and Belt of Cementation.*

Under the action of atmospheric agencies the rocks at and near the surface suffer loss of their materials and waste away. This process may be described as weathering. Deeper in the earth, the materials derived from the upper rocks are largely redeposited. This process is one of cementation. The belt of weathering and the belt of cementation are not separated by a sharp line; and, moreover, with the processes of erosion the lower limit of the activity of the atmospheric agencies has constantly migrated downward. These belts are related to the topography of the country, the plane separating them being higher in the hills than in the valleys. Consequently, during the long period which has been required for the removal of the stratified rocks to their present limits, there has been a shifting

downward and to the southward, as the streams have cut their valleys deeper and the escarpments have retreated to the southward. In the northern part of the field the belt of weathering, which was formerly in the Mississippian limestones, has, since the removal of these beds, reached the Ordovician rocks. To the southward, as a result of the rugged topography, it lies partly in the Ordovician and partly in the Mississippian. At the base of the Boston mountains, where the Mississippian limestones have been but recently uncovered, it has descended but a short distance into the upper portion of this formation. The rocks in the northern portion of the zinc- and lead-district of Northern Arkansas may, accordingly, be considered as exhibiting the more advanced stages of the processes of weathering and erosion.

*Source of the Ores.*

It is generally accepted that the zinc- and lead-deposits of this region have been accumulated by the action of circulating-waters which have dissolved the broadly disseminated ores in the limestones of the region, and carried them in solution to certain places where the conditions were favorable for their re-deposition. A study of the nature of the ores and their gangue materials, and of the geological history of the region, makes it apparent that at least the latest concentration of the ores in the Ordovician has been largely the result of downward and lateral movements. The metallic sulphides may have been mainly derived from the Mississippian limestones which formerly had a wider distribution, and from the Ordovician, or from both formations. An examination of the Mississippian rocks shows that they have been leached by surface-waters. Where they are exposed in railway cuts they exhibit decay to considerable depths, and within the area of their outcrop there are numerous sink-holes in which the water disappears into underground channels. The surface-cherts which have been derived from the weathering of those rocks are frequently porous and spongy, thus indicating the loss of silica. Secondary silica is not infrequently a gangue of the ores in the Ordovician formation. The Mississippian limestones contain notable deposits of zinc and lead at many localities in the Ozark region, and where there is ore in the Ordovician the Mississippian limestones have formerly overlaid the area. The mines of Southwestern Missouri

around Joplin are in this formation, and in Northern Arkansas, as has already been stated, prospecting has shown that in the portion of the district where they have been but recently exposed to the action of surface-waters, probably as a result of a first concentration, they carry considerable lead and zinc.

### *Classification of the Ore-Deposits.*

The most important deposits of the district are the sulphide ores of lead and zinc, or, as they are commonly called, galena and blende. In the Ordovician dolomites there are two principal classes of these deposits, which are characterized by the gangue material. One class is distinguished by their occurrence with secondary chert, which occurs as a siliceous replacement of the dolomites, or filling fractures in these rocks; in the other, there is associated with the ore a large amount of dolomite spar, which forms a cementing material in the breccias. In certain of the mines there is, in addition to these main ore-bodies, accessory ore which replaces the country-rock to some extent adjacent to the main ore-body, without the development of secondary chert or spar.

In the Mississippian limestones the primary ore-deposits are accompanied by secondary chert and calcite. They are related to fractures, and in some instances to fault-planes. In the latter case they usually occupy breccias. Accessory ore replacing the country-rock is sometimes present with these deposits.

The Northern Arkansas field contains important deposits of oxidized ores. These are the carbonates and silicates. They are derived from the primary sulphides, and are due to the alteration of the sulphides by the action of surface-waters. In discussing the genesis of the ores, the important problem is the origin of the sulphide deposits, the relation of the oxidized deposits to the sulphide deposits being clearly evident.

*Primary Deposition of the Sulphide Ores.*—The action of ground-waters in the belt of weathering, and to a considerable extent in the belt of cementation, resulted in the solution and transportation of the ores. As the water percolated downward and moved laterally (and perhaps later upward), it reached places where deposition occurred. In the early part of the journey of the waters, through the action of the carbon dioxide

and the humic acids, notable quantities of silica, as well as the ores, were taken into solution. In the latter part of the journey these waters dissolved the lime and magnesium carbonate and deposited silica and sulphides; the resulting ores were supposedly transported as sulphates. The reduction of the metals to sulphides was probably accomplished through the agency of organic matter and pyrite in the rocks, directly or indirectly, and deposition of the sulphides occurred along with the formation of the secondary chert and spar.

The superposition of the original cherts in the Mississippian limestones, and the occurrence of the secondary cherts and spar in the dolomites, are entirely in accordance with this theory.

*Deposition of the Oxidized Ores.*—The oxidized ores of Northern Arkansas are the carbonates and silicates, which have been derived from the sulphide ore-bodies. They are, accordingly, relatively later, and have been produced since erosion has brought the sulphides into the zone of weathering. The descending waters, carrying carbon dioxide, have transformed the blende and galena. In some cases redeposition has taken place immediately, and not infrequently oxidized ores are found as incrustations on the sulphides. In other cases they are found along water-channels, or in the open spaces and on the surfaces of the country-rock. In the exposed faces of ore-bearing beds and the upper portions of workings, secondary ores often predominate. When mining operations are carried into the rocks that are under cover or have been protected from the action of ground-waters, the carbonates and silicates decrease, and galena and blende are found to be the predominating ores.

*Secondary Deposition of the Sulphides.*—The sulphide ores, which were dissolved by descending waters, have not all been redeposited within the belt of weathering. Such portions as were retained in solution, upon reaching the belt of cementation, were redeposited as sulphides, the processes in this case being the same as in the primary deposition, and the ore-bodies belonging to a second generation. In the lower horizons of the Ordovician dolomites considerable zinc-ore is found which occurs as bright, clean crystals associated with drusy quartz or in openings formed by fracture. Such deposits are usually lean, and thus far no workable body of ore of this nature has been discovered.

Observations have not shown that there is a criterion for clearly distinguishing the secondary sulphide ores, which may have originated by migration, from the primary deposits. It is not improbable that the solutions at the time of the first concentration may have deposited most of their ore in the upper horizons, in which case the deeper deposits would have the characteristics above described and assigned to the ores of the second generation.

*Sulphide Deposits Associated with Secondary Chert.*—Where chert is the principal gangue of the blende and galena, deposition in the Ordovician has taken place by the replacement of the dolomites and the filling of fracture-spaces and cementation of breccias. Secondary chert, when freshly exposed, usually has a bluish color. It may be distinguished from the other country-rocks by means of its hardness, since it cannot be scratched with a knife. It frequently has a banded or bedded appearance, which corresponds to the bedding of the original dolomite, and crystals of ore, usually well formed and distinct, occur within the mass. In case some of the sulphides have been leached out, moulds of the blende are seen, which gives the chert a honeycombed appearance. The richer deposits appear to be related to fracture-zones, and occur along the fissures and replacing the adjacent beds. The path of the ore-bearing solutions, in descending, has apparently been along fractures and laterally along the bedding-planes, and the mineralization decreases away from the fracture-zone.

*Sulphide Deposits in Bedded Breccias.*—In the brecciated beds of the Ordovician dolomites, the open spaces between the fragments have afforded channels for the ore-bearing solutions, and the precipitation of the sulphides and dolomite or pink spar has usually taken place without dissolving the country-rock to any appreciable extent. The pink spar is not always accompanied by ore. The sulphides have been deposited in a somewhat local way, many factors being concerned. Not infrequently, in prospecting the breccias containing pink spar, but little ore is found; and when the ore-bearing breccias are followed for a considerable distance, they usually show a decrease in the amount of ore.

*Sulphide Deposits in Fissures.*—Most of the mines and prospects in the Mississippian limestones are related to fissures.

The ore occurs in the material which fills the fissures or in the fissure and the openings adjacent to it. These deposits differ from those in the fractured dolomites in being more clearly defined. The gangue is usually secondary chert and calcite. The walls of the fissures exhibit slickensides as a result of the movement of beds, and frequently indicate displacement in a horizontal direction.

*Sulphide Deposits in Fault-Breccias.*—Where the Mississippian limestones have been displaced by normal faulting, and the rocks have been dragged, they frequently exhibit brecciation. The angular fragments are largely primary chert, and the ore occurs associated with a calcareous and siliceous matrix, which cements the breccia.

*Sulphide Deposits in Country-Rock.*—In many of the mines and prospects the country-rock has not been mineralized. In other cases, for a short distance adjacent to the fissures, fractures and water-channels, the ore-bearing solutions have formed what is here called accessory ore. The action in this place has been one of replacement. The country-rock exhibits recrystallization, and carries small crystals of ore. Where accessory ore is found, the main ore-body is usually rich, and there is a suggestion that deposition in the country-rock resulted because of the large amount of ore in solution at these places. In the Northern Arkansas district, the scattered crystals of blende in the country-rock are spoken of as disseminated ore. This term, unfortunately, is not quite appropriate, and, accordingly, the word "*accessory*" is suggested, since it does not imply the mode of deposition usually ascribed to disseminated ores. Accessory ore, inasmuch as it is usually found associated with rich ore-bodies, is looked upon by the prospectors as a favorable indication. The ore in secondary chert is not included under this head.

*Ore Associated with Quartz-Druses.*—Frequently, in the Ordovician dolomites, the lower ore-horizons exhibit quartz-druses and surfaces covered with minute quartz-crystals. The ore in these rocks occurs as clean, bright crystals deposited on the quartz. It is found only occasionally in large masses, but is generally distributed through the rocks. The probability is that it represents a migration from the higher horizons, but it is possible that it was deposited from the depleted solutions at the time of primary deposition.



*Opinions of Previous Writers.\**—Certain ideas have been advanced in the previous reports on the Northern Arkansas field which are not accepted by the writer. These largely pertain to theoretical considerations, although some of them deal with the geological facts. There is no opportunity in this article for a discussion of the differences of opinion which have arisen, but it is thought best to mention certain points which are obvious from a review of the literature.

The writer has argued that the bedded breccias were produced by the movement of one stratum past another, as a result of compressive forces. Mr. Branner, in speaking of the breccias, says that the bedded breccias were not formed on fractures, but along ancient underground water-courses. The breccia-deposits which Mr. Bain described were considered by him to have been formed along zones of pressure. He speaks of limestone conglomerates, which, in my opinion, are really breccias. The relation of these so-called conglomerates to the brecciated beds, and the fact of extensive brecciation due to differential horizontal movement, do not seem to have been recognized by Mr. Bain.

My interpretation of the faulting of the region is decidedly opposed to that of Mr. Branner. I consider certain of the most important faults to be normal, and some of these were described and figured by Mr. Branner as thrust-faults.

In regard to the theory of ore-deposition, all the differences cannot be pointed out here. However, Mr. Branner described bedded deposits contemporaneous with the rocks in which they occur. Such ore-bodies, I believe, are the result of secondary alteration and replacement. Mr. Bain has given considerable prominence to what he calls disseminated ores in compact limestone and unbroken conglomerate. This description does not seem to be a correct characterization of any of the main ore-bodies of the district.

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\* "The Zinc- and Lead-Region of North Arkansas," by J. C. Branner, *Ann. Rept. Ark. Geol. Survey for 1892* (published in 1900), vol. v.; also *Trans.*, xxxi., p. 572. "Preliminary Report on the Lead- and Zinc-Deposits of the Ozark Region," by H. F. Bain, *Twenty-Second Ann. Rept. U. S. Geol. Survey*, Pt. II., 1902, pp. 195-202. "Lead- and Zinc-Deposits of the Mississippi Valley," by C. R. Van Hise and H. Foster Bain, read before the Institution of Mining Engineers, London, May 29, 1902, excerpt from *Trans. Inst. Min. Engrs.*, pp. 34, 35.

## Electrolytic Lead-Refining.

BY ANSON G. BETTS, TROY, N. Y.

(Albany Meeting, February, 1903.)

A SOLUTION of lead-fluosilicate, containing an excess of fluosilicic acid, has been found to work very satisfactorily as an electrolyte for refining lead. It conducts the current well, is easily handled and stored, non-volatile and stable under electrolysis, may be made to contain a considerable amount of dissolved lead, and is easily prepared from inexpensive materials. It possesses, however, in common with other lead electrolytes, the defect of yielding a deposit of lead lacking in solidity, which grows in crystalline branches toward the anodes, causing short circuits. But if a reducing-action (practically accomplished by the addition of gelatine or glue) be given to the solution, a perfectly solid and dense deposit is obtained, having very nearly the same structure as electrolytically-deposited copper, and a specific gravity of about 11.36—that of cast-lead.

Lead-fluosilicate may be crystallized in very soluble, brilliant crystals, resembling those of lead-nitrate and containing four molecules of water of crystallization, with the formula  $\text{PbSiF}_6 \cdot 4\text{H}_2\text{O}$ . This salt dissolves at  $15^\circ \text{C}$ . in 28 per cent. of its weight of water, making a sirupy solution of 2.38 sp. gr. Heated to  $60^\circ \text{C}$ ., it melts in its water of crystallization. A neutral solution of lead-fluosilicate is partially decomposed on heating, with the formation of a basic insoluble salt and free fluosilicic acid, which keeps the rest of the salt in solution. This decomposition ends when the solution contains, perhaps, 2 per cent. of free acid; and the solution may then be evaporated without further decomposition. The solutions desired for refining are not liable to this decomposition, since they contain much more than 2 per cent. of free acid. The electrical conductivity depends mainly on the acidity of the solution.

My first experiments were carried out without the addition of gelatine to the fluosilicate solution. The lead-deposit con-

sisted of more or less separate crystals that grew toward the anode, and, finally, caused short-circuits. The cathodes, which were sheet-iron plates, lead-plated and paraffined, had to be removed periodically from the tanks and passed through rolls, to pack down the lead. When gelatine has been added in small quantities, the density of the lead is greater than can be produced by rolling the crystalline deposit, unless great pressure is used.

The Canadian Smelting-Works, Trail, British Columbia, have installed a refinery, making use of this process. There are 28 refining-tanks, each 86 in. long, 30 in. wide and 42 in. deep, and each receiving 22 anodes of lead-bullion with an area of 26 by 33 in. exposed to the electrolyte on each side, and 23 cathodes of sheet-lead, about  $\frac{1}{16}$  in. thick, prepared by deposition on lead-plated and paraffined-iron cathodes. The cathodes are suspended from 0.5 by 1 in. copper-bars, resting crosswise on the sides of the tanks. The experiment has been thoroughly tried, of using iron-sheets to receive a deposit thicker than  $\frac{1}{16}$  in.; that is, suitable for direct melting without the necessity of increasing its weight by further deposition as an independent cathode; but the iron-sheets are expensive, and are slowly pitted by the action of the acid-solution; and the lead-deposits thus obtained are much less smooth and pure than those on lead-sheets.

The smoothness and the purity of the deposited lead are proportional. Most of the impurity seems to be introduced mechanically through the attachment of floating particles of slime to irregularities on the cathodes. The effect of roughness is cumulative: it is often observed that particles of slime attract an undue amount of current, resulting in the lumps seen on the cathodes. Samples taken at the same time showed from 1 to 2.5 oz. silver per ton in rough pieces from the iron cathodes, 0.25 oz. as an average for the lead-sheet cathodes, and only 0.04 oz. in samples selected for their smoothness. The variation in the amount of silver (which is determined frequently) in the samples of refined-lead is attributed not to the greater or less turbidity of the electrolyte at different times, but to the employment of new men in the refinery, who require some experience before they remove cathodes without detaching some slime from the neighboring anodes.

The shape of the electrodes, and the method of handling them to and from the tanks, are shown in Fig. 1.

Each tank is capable of yielding, with a current of 4,000 amperes, 750 lb. of refined-lead per day. The voltage required to pass this current was higher than expected, as explained below; and for this reason, and also because the losses of solution were very heavy until proper apparatus was put in to wash thoroughly the large volume of slime produced (resulting in a weakened electrolyte), the current used has probably averaged about 3,000 amperes. The short-circuits were also troublesome, though this difficulty has been greatly reduced by frequent inspection and careful placing of the electrodes. At one time, the solution in use had the following composition in grammes per 100 c.c.: Pb, 6.07; Sb, 0.0192; Fe, 0.2490;  $\text{SiF}_6$ , 6.93, and As, a trace. The current passing was 2,800 amperes, with an average of about 0.44 volts per tank, including bus-bars and contacts. It is not known what was the loss of efficiency on that date, due to short-circuits; and it is, therefore, impossible to say what resistance this electrolyte constituted.

Hydrofluoric acid of 35 per cent., used as a starting-material for the preparation of the electrolyte, is run by gravity through a series of tanks for conversion into lead-fluosilicate. In the top tank is a layer of quartz 2 ft. thick, in passing through which the hydrofluoric acid dissolves silica, forming fluosilicic acid. White-lead (lead-carbonate) in the required quantity is added in the next tank, where it dissolves readily and completely with effervescence. All sulphuric acid and any hydrofluoric acid that may not have reacted with silica settle out in combination with lead as lead-sulphate and lead-fluoride. Lead-fluosilicate is one of the most soluble of salts; so there is never any danger of its crystallizing out at any degree of concentration possible under this method. The lead-solution is then filtered and run by gravity into the refining-tanks.

The solution originally used at Trail contained about 6 per cent. Pb and 15 per cent.  $\text{SiF}_6$ .

The electrical resistance in the tanks was found to be greater than had been calculated for the same solution, plus an allowance for loss of voltage in the contacts and conductors. This is partly, at least, due to the resistance to free motion of the

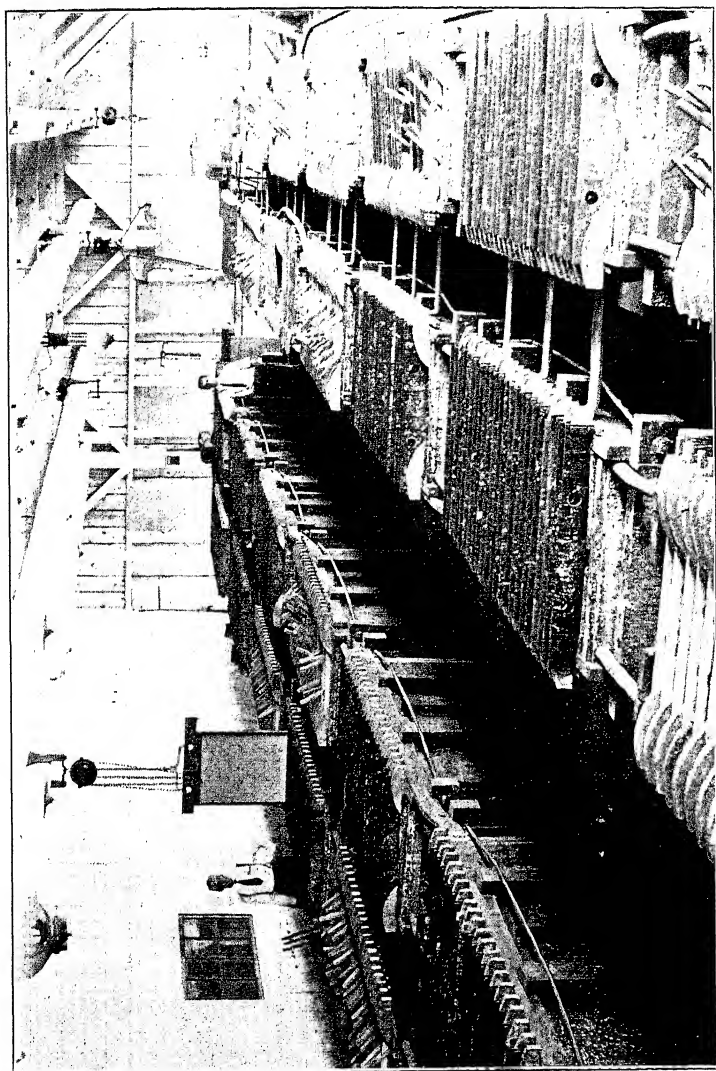
electrolyte, in the neighborhood of the anode, offered by a layer of slime which may be anything up to 0.5 in. thick. During electrolysis, the  $\text{SiF}_6$  ions travel toward the anodes, and there combine with lead. The lead and hydrogen travel in the opposite direction and out of the slime; but there are comparatively few lead ions present, so that the solution in the neighborhood of the anodes must increase in concentration and tend to become neutral. This greater concentration causes an E. M. F. of polarization to act against the E. M. F. of the dynamo. This amounted to about 0.02 v. for each tank. The greater effect comes from the greater resistance of the neutral solution with which the slime is saturated. There is, consequently, an advantage in working with rather thin anodes, when the bullion is impure enough to leave slime sticking to the plates. A compensating advantage is found in the increased ease of removing the slime with the anodes, and wiping it off the scrap in special tanks, instead of emptying the tanks and cleaning out, as is done in copper-refineries.

It is very necessary to have adequate apparatus for washing solution out of the slime. The filter first used consisted of a supported filtering-cloth with suction underneath. It was very difficult to get this to do satisfactory work by reason of the large amount of fluosilicate to be washed out with only a limited amount of water. At the present time the slime is first stirred up with the ordinary electrolyte several times, and allowed to settle, before starting to wash with water at all. The Trail plant produces daily 8 or 10 cu. ft. of anode residue, of which over 90 per cent. by volume is solution. The evaporation from the total tank-surface of something like 400 sq. ft. is only about 15 cu. ft. daily; so that only a limited amount of wash-water is to be used—namely, enough to replace the evaporated water, plus the volume of the slime taken out.

The tanks are made of 2-in. cedar, bolted together and thoroughly painted with rubber-paint. Any leaks are caught underneath on sloping-boards. Solution is circulated from one tank to another by gravity, and is pumped from the lowest to the highest by means of a wooden pump. The 22 anodes in each tank together weigh about 3 tons, and dissolve in from 8 to 10 days, two sets of cathodes usually being used with each set of anodes. While 300-lb. cathodes can be made, the short-

circuiting gets so troublesome with the spacing used that the loss of capacity is more disadvantageous than the extra work of putting in and taking out more plates. The lead-sheets used for cathodes are made by depositing about  $\frac{1}{16}$  in. metal on par-

FIG. 1.



Electrolytic Lead-Refinery, Canadian Smelting-Works, Trail, B. C.

affined steel-sheets in four of the tanks, which are different from the others only in being a little deeper.

The anodes may contain any or all of the elements, gold, silver, copper, tin, antimony, arsenic, bismuth, cadmium, zinc,

iron, nickel, cobalt and sulphur. It would be expected that gold, silver, copper, antimony, arsenic and bismuth, being more electronegative than lead, would remain in the slime in the metallic state, with, perhaps, tin, while iron, zinc, nickel and cobalt would dissolve. It appears that tin stands in the same relation to lead that nickel does to iron, that is, they have about the same electromotive forces of solution, with the consequence that they can behave as one metal and dissolve and deposit together. Iron, contrary to expectation, dissolves only slightly, while the slime will carry about 1 per cent. of it. It appears from this that the iron exists in the lead in the form of matte. Arsenic, antimony, bismuth, and copper have electromotive forces of solution more than 0.3 volt below that of lead. As there is no chance that any particle of one of these impurities will have an electric potential of 0.3 volt above that of the lead with which it is in metallic contact, there is no chance that they will be dissolved by the action of the current. The same is even more certainly true of silver and gold. The behavior of bismuth is interesting and satisfactory. It is as completely removed by this process of refining as antimony is. No other process of refining lead will remove this objectionable impurity so completely. Tin has been found in the refined-lead to the extent of 0.02 per cent. This we had no difficulty in removing from the lead by poling before casting. There is always a certain amount of dross formed in melting down the cathodes; and the lead-oxide of this reacts with the tin in the lead at a comparatively low temperature.

The extra amount of dross formed in poling is small, and amounts to less than 1 per cent. of the lead. The dross carries more antimony and arsenic than the lead, as well as all the tin. The total amount of dross formed is about 4 per cent. Table I. shows its composition.

TABLE I.—*Analyses of Dross.*

For analyses of the lead from which this dross was taken, see Table II.

No.	No. in Table II.	Cu. Per Cent.	As. Per Cent.	Sb. Per Cent.	Fe. Per Cent.	Zn. Per Ct.
1	2	0.0005	0.0003	0.0016	0.0016	none
2	3	0.0010	0.0008	0.0107	0.0011	"

The electrolyte takes up no impurities, except, possibly, a small part of the iron and zinc. Estimating that the anodes contain 0.01 per cent. of zinc and soluble iron, and that there are 150 cu. ft. of the solution in the refinery for every ton of lead turned out daily, in one year the 150 cu. ft. will have taken up 72 lb. of iron and zinc, or about 1 per cent. These impurities can accumulate to a much greater extent than this before their presence will become objectionable. It is possible to purify the electrolyte in several ways. For example, the lead can be removed by precipitation with sulphuric acid, and the fluosilicic acid precipitated with salt as sodium fluosilicate. By distillation with sulphuric acid the fluosilicic acid could be recovered, this process, theoretically, requiring but one-third as much sulphuric acid as the decomposition of fluorspar, in which the fluorine was originally contained.

The only danger of lead-poisoning to which the workmen are exposed occurs in melting the lead and casting it. In this respect the electrolytic process presents a distinct sanitary advance.

For the treatment of slime, the only method in general use consists in suspending the slime in a solution capable of dissolving the impurities and supplying, by a jet of steam and air forced into the solution, the air necessary for its reaction with, and solution of, such an inactive metal as copper. After the impurities have been mostly dissolved, the slime is filtered off, dried and melted, under such fluxes as soda, to a doré bullion.

The amount of power required is calculated thus: Five amperes in 24 hours make 1 lb. of lead per tank. One ton of lead equals 10,000 ampere-days, and at 0.35 volts per tank, 3,500 watt-days, or 4.7 E.H.P.-days. Allowing 10 per cent. loss of efficiency in the tanks (we always get less lead than the current which is passing would indicate), and of 8 per cent. loss in the generator increases this to about 5.6 H.P.-days, and a further allowance for the electric-lights and other applications gives from 7 to 8 H.P.-days as about the amount per ton of lead. At \$30 per year, this item of cost is something like 65 cents per ton of lead. So this is an electro-chemical process not especially favored by water-power.



The cost of labor is not greater than in the zinc-desilverization process. A comparison between this process and the Parkes process, on the assumption that the costs for labor, interest and general expenses are about equal, shows that about \$1 worth of zinc and a considerable amount of coal and coke have been done away with, at the expense of power, equal to about 175 H.P.-hours., of the average value of perhaps 65 cents, and a small amount of coal for melting the lead in the electrolytic method.

More important, however, is the greater saving of the metal-values by reason of increased yields of gold, silver, lead, antimony and bismuth, and the freedom of the refined-lead from bismuth.

Tables II., III. and IV. show the composition of bullion, slimes and refined lead.

Tables V., VI., VII. and VIII. give the results obtained experimentally in the laboratory on lots of a few pounds up to a few hundred pounds. The results in Tables VI. and VII. were given me by the companies for which the experiments were made.

TABLE II.—*Analyses of Bullion.*

No.	Fe. Per Cent.	Cu. Per Cent.	Sb. Per Cent.	Sn. Per Cent.	As. Per Cent.	Ag. Per Cent.	Au. Per Cent.	Pb. Per Cent.	Ag. Oz. p. T.	Au. Oz. p. T.
1	0.0075	0.1700	0.5400	0.0118	0.1460	1.0962	0.0085	98.0200	319.7	2.49
2	0.0115	0.1500	0.6100	0.0158	0.0960	1.2014	0.0086	97.9068	350.4	2.52
3	0.0070	0.1600	0.4000	0.0474	0.1330	1.0738	0.0123	98.1665	313.2	3.6
4	0.0165	0.1400	0.7000	0.0236	0.3120	0.8914	0.0151	97.9014	260.0	4.42
5	0.0120	0.1400	0.8700	0.0432	0.2250	0.6082	0.0124	98.0882	177.4	3.63
6	0.0055	0.1300	0.7300	0.0816	0.1030	0.6600	0.0106	98.2693	192.5	3.10
7	0.0380	0.3600	0.4030	.....	tr.	0.7230	0.0180	98.4580	210.9	5.25

TABLE III.—*Analyses of Slimes.*

Fe. Per Ct.	Cu. Per Ct.	Sb. Per Ct.	As. Per Ct.	Ag. Per Ct.	Pb. Per Ct.	Zn. Per Ct.	Bi. Per Ct.
1.27	8.83	27.10	12.42	28.15	17.05	none	none
1.12	22.36	21.16	5.40	23.05	10.62	"	"

TABLE IV.—*Analyses of Refined Lead.*

No.	Cu. Per Cent.	As. Per Cent.	Sb. Per Cent.	Fe. Per Cent.	Zn Per Cent.	Sn. Per Cent.	Ag. Oz. P. T.	Ni, Co, Cd. Per Cent.	Bi. Per Cent.
1	0.0006	0.0008	0.0005	0.0010	none	.....	.....	.....	.....
2	0.0003	0.0002	0.0010	0.0010	"	.....	0.24	.....	.....
3	0.0009	0.0001	0.0009	0.0008	.....	.....	0.47	none	.....
4	0.0016	.....	0.0017	0.0014	.....	.....	0.22	.....	.....
5	0.0003	.....	0.0060	0.0003	.....	.....	0.22	none	.....
6	0.0020	.....	0.0010	0.0046	.....	.....	0.14	.....	.....
7	0.0004	none	0.0066	0.0013	none	0.0035	0.25	.....	.....
8	0.0004	.....	0.0038	0.0004	"	0.0035	0.28	.....	.....
9	0.0005	.....	0.0052	0.0004	"	0.0039	0.43	.....	.....
10	0.0003	none	0.0060	0.0003	"	0.0049	0.32	.....	.....
11	0.0003	"	0.0042	0.0013	"	0.0059	0.22	.....	.....
12	0.0005	"	0.0055	0.0009	"	0.0049	0.11	.....	.....
13	0.0005	"	0.0055	0.0007	"	0.0091	0.14	.....	.....
14	0.0004	"	0.0063	0.0005	"	0.0012	0.24	.....	.....
15	0.0003	"	0.0072	0.0003	"	0.0024	0.22	.....	.....
16	0.0006	"	0.0062	0.0012	"	0.0083	0.38	.....	.....
17	0.0006	"	0.0072	0.0011	.....	0.0080	0.22	.....	.....
18	0.0006	"	0.0057	0.0010	.....	0.0053	0.34	.....	.....
19	0.0005	"	0.0066	0.0016	.....	0.0140	0.22	.....	.....
20	0.0004	"	0.0047	0.0015	.....	0.0072	0.23	.....	.....
21	0.0004	"	0.0034	0.0016	.....	trace	0.38	none	none
21	0.0022	"	0.0010	0.0046	none	0.0081	.....	.....	.....

TABLE V.—*Analyses of Bullion and Refined Lead.*

	Ag. Per Cent.	Cu. Per Cent.	Sb. Per Cent.	Pb. Per Cent.
Bullion, . . . . .	0.50	0.31	0.43	98.76
Refined lead, . . . . .	0.0003	0.0007	0.0019	99.9971

TABLE VI.—*Analyses of Bullion and Refined Lead.*

	Cu. Per Ct.	Bi. Per Ct.	As. Per Ct.	Sb. Per Ct.	Ag. Oz. p. T.	Ag. Per Ct.	Au. Oz. p. T.	Fe. Per Ct.	Zn. Per Ct.
Bullion, . . . . .	0.75	1.22	0.936	0.6832	358.89	.....	1.71	.....	.....
Refined lead, 0.0027	.....	.0087	0.0025	0.0000	.....	0.0010	none	0.0022	0.0018

TABLE VII.—*Analyses of Bullion, Refined Lead and Slimes.*

	Pb. Per Ct.	Cu. Per Ct.	As. Per Ct.	Sb. Per Ct.	Ag. Oz. per T.	Ag. Per Ct.	Fe, Zn, Ni, Co. Per Ct.	Bi.
Bullion, . . . . .	96.73	0.096	0.85	1.42	about 275†	.....	.....	.....
Refined lead, . . . . .	.....	0.0013	0.00506	0.0028	.....	0.00068	0.0027	trace
Slimes (dry sample), . . . . .	9.05	1.9	9.14	29.51	9866.9	.....	0.49	trace

TABLE VIII.—*Analyses of Bullion, Refined Lead and Slimes.*

	Pb. Per Cent.	Cu. Per Cent.	Bi. Per Cent.	Ag. Per Cent.	Sb. Per Cent.	As. Per Cent.
Bullion, . . . . .	87.14	1.40	0.14	0.64	4.0	7.4
Lead, . . . . .	.....	0.0010	0.0022	.....	0.0017	trace
Slimes, . . . . .	10.3	9.3	0.52	4.7	25.32	44.58

\* Poled. .

† Silver not given. This was the case, also, with the gold in the bullion. The slimes contained 0.131 per cent. of gold, or 39.1 oz. per ton.

The process here described has been patented in the United States, Canada, Mexico, Belgium, France, Great Britain, Italy, Spain, New South Wales, Victoria and South Australia.

The success thus attained in the electrolysis of lead, generally accepted hitherto as impracticable, may give some encouragement to the employment of similar methods in the treatment of some of the other metals, especially as it is shown to be possible to apply simple means to obviate the chief trouble, spongy deposits.

I wish to express my thanks to Mr. W. H. Aldridge, Dr. Edward F. Kern, and Dr. William Valentine, of the Canadian Smelting-Works, for their kindness in giving information and analyses.

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### A Test for Precious Metals in Cyanide Solutions.

BY ALBERT ARENTS, ALAMEDA, CALIFORNIA.

(Albany Meeting, February, 1903.)

THIS test is based upon the fact that metallic copper will precipitate gold and silver upon its surface from acid solutions. Of course, the fact is not new, but its application is probably so. I have used the method with success; it recommends itself by the rapidity and ease with which it may be carried on.

An auriferous cyanide solution, if made acid with sulphuric acid and boiled with finely divided, pulverulent, metallic copper, will, within a short time, deposit its gold-content on the copper. Any silver in the solution is also precipitated. If this mixture is now filtered, the filter and contents may at once be subjected to a crucible assay treatment, and its lead buttons cupelled and determined.

If, instead of taking cement-copper, or any metallic copper-powder, a solution of bluestone is used after acidification, and a few small pieces of sheet aluminum are added, and the solution boiled until all the copper has come down, the result as to the precipitation of gold and silver is the same. This modification takes more time and attention in boiling. If aluminum has been used, it should go into the crucible with the filter and

its contents. Commercial cement-copper is particularly fitted for this test, because the acid, in taking up any basic iron- or copper-salts of the cement-copper, renders the copper as finely divided as it is customary to obtain it in the sluice-boxes of "copper leachers." The finer and the more pulverulent the copper is, the greater is its surface and the more energetic the precipitation, thus permitting a minimum amount of copper to be used.

In applying the method, I use, as a rule, 250 c.c. of the solution to be tested; add a few c.c. of sulphuric acid; agitate for several seconds, and then add not less (although not much more) than one gramme of cement-copper. Now follows heating to boiling. This is kept up for about 10 minutes, so that the rising steam-bubbles keep the mixture well agitated. The mixture is then filtered through a 7-inch-diameter gray filter-paper. No washing is done. As soon as the filtering is finished, one-third of a crucible-charge of flux is added to the filter containing all the sediment of the mixture. Some of the moisture is rapidly absorbed by the flux, which permits the folding of the filter's rim upon the charge and its subsequent removal without loss or tearing. One-third of a crucible-charge of flux having previously been placed upon the bottom of the crucible which is to be used for melting, the filter is transferred to the crucible, well tucked down, and the last one-third of the crucible-charge is placed on top of the filter in the crucible. It is then ready for the furnace. The filter itself furnishes the reducing-agent for the assay. I use 30 grammes litharge and the usual amount of borax and soda, employing a No. F crucible for melting. About 20 grammes of lead are obtained. The lead button comes out bright and clean, and upon cupelling furnishes a bead free from copper.

Possibly this method of testing for gold and silver may be used upon other solutions than cyanide; also, for solutions from testing metallic copper for precious metals, when the solutions do not contain nitric acid in any form.

## Biographical Notice of Abram S. Hewitt.

BY R. W. RAYMOND, NEW YORK CITY.

(Albany Meeting, February, 1903.)

THE tidings of Mr. Hewitt's death, cabled to me at Rome, Italy, brought me a great shock as well as a sincere sorrow. I had left him, a few weeks before, not indeed in vigorous health, but still quite able to command recognition as a continued power in public affairs; and I thought him destined to enjoy for many years that final stage of a long, active and honored career, in which, having passed beyond the struggles and rivalries of business and of politics, his life and character could be justly appreciated by all men, like some noble work of art, already crowned, and marked henceforward "*hors de concours*." Moreover, he was, and would be to the end, not merely the passive recipient of admiration and esteem, but the source of inspiring appeals, effective warnings and wise counsel. His rich and varied experience he had placed at the service of his fellows. His lightest word was weighty, and those who contradicted him only injured their own cause thereby. Great trusts (true "trusts," in the old and proper sense of the term), like the Cooper Union, the Carnegie Institution, and others, still commanded his disinterested services. Great principles of liberty and of justice continued to be reinforced in perilous crises by his powerful advocacy; great measures were daily receiving from him a decisive impetus. No wonder it seemed to us that this happy period of reward so well earned, and of beneficence so welcome to innumerable recipients of its blessing, must continue yet awhile. We did not expect the night instantly to engulf the full-orbed sun. Nor, indeed, has this been our experience; in this instance, the sunset has left an afterglow more luminous than many a noonday.

For the principal data contained in the following brief sketch of Mr. Hewitt's life, I rely upon memoranda personally approved by him, some months before his death, for my use in

the preparation of such a sketch. I need not say that they furnish but a scanty outline and summary of a singularly active and fruitful career.

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In the last decade of the eighteenth century, four chosen workmen were sent to this country from the famous Soho works of Boulton & Watt, to set up, at the Centre Square water-works in Philadelphia, the first steam-engine ever seen on this side of the Atlantic. One of the four entrusted with this novel and weighty responsibility was a youth of eighteen, named John Hewitt, who remained in the United States after discharging this duty, and a little later (in 1793-94), at the small machine-shop attached to the Schuyler copper-mine, near Belleville, N. J., served as pattern-maker in the construction of the first steam-engine wholly built on this continent.\*

Deciding to risk his fortunes with the new republic, the young English mechanic settled in the State of New York, and took a wife from a French Huguenot family, bearing the name of Garnier, which, in the modified form of Gurnee, is now well known in the business and social circles of Manhattan. Of this parentage, Abram Stevens Hewitt was born, July 31, 1822, at Haverstraw, Rockland county, N. Y.

He began his education in the public schools of New York City, where he won, as the result of a competitive examination, a scholarship in Columbia College. Throughout his college-course he stood at the head of his class, and with that rank he was graduated in 1842. Although the scholarship which he had earned relieved him from dues for tuition, he was obliged to support himself, in other respects, by extra work as a teacher; and at the time of his graduation he was already, though but twenty years of age, a regular tutor of mathematics in the college. It was in this capacity that he made the acquaintance of Edward, the only son of Peter Cooper.

For a year after his graduation he held the position of acting-professor of mathematics at Columbia, pursuing, at the same time, the study of law, and practicing such economy that, at the end of that period, he had managed to save about \$1000,

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\* See "A Century of Mining and Metallurgy in the United States" (Presidential Address by Abram S. Hewitt), *Trans.*, v., 168.

which enabled him, in 1844, to make a trip to Europe, in company with young Cooper. On the return-voyage, from Leghorn to Mobile, the ship in which they were the only passengers was wrecked not far from Cape May. The story of their wreck and rescue is a thrilling one, but cannot be told at length in this sketch, which must neglect so many other events of greater public significance. Many fanciful exaggerations of it have been circulated from time to time, to the amusement or annoyance of the parties concerned. The essential facts are, that the voyagers escaped in two boats, the captain generously assigning to the mate and his party of five seamen the smaller and safer boat, while he himself, with nine of the crew and the two passengers (who chose to go with him), took a boat so decayed and dilapidated that they dared not step on the bottom for fear of going through, or carry any load beyond the clothes they wore, or even strain the frail structure by rowing. The mate's boat soon encountered the ship *Atalanta*, commanded by Captain George B. Raymond, which was lying-to, partially disabled by the gale. Captain Raymond, after taking this party on board, began to cruise vigilantly for the other. It was nearly dark when his ship was seen from the boat; and two or three hours passed before the almost despairing sailors, by incessant shouting in relays, made their whereabouts known, and were rescued, after twelve hours' drifting. In 1886, referring to this experience, Mr. Hewitt said:

"I landed at New York in midwinter, in a borrowed suit of sailor's clothing, and I had three silver dollars in my pocket, my entire worldly wealth.

"I was then twenty-two years old, and that 'accident' was the turning-point of my life. It taught me for the first time that I could stand in the face of death without fear and without flinching. It taught me another thing—that my life, which had been miraculously rescued, belonged not to me; and from that hour I gave it to the work which from that time has been in my thoughts—the welfare of my fellow-citizens. For thirty years I have never turned aside from that work. The task which I had set for myself was to contribute, as far as I could, to the employment of men, so that they could help themselves, and not be made the subjects of public charity. Self-help is the remedy for all the evils of which men complain. I have had to help myself from the earliest year I can remember; and every struggling young man who chooses to follow the same rule, who will help himself and not become dependent on public or private charity, can achieve a measure of success that will satisfy every independent citizen."

This common experience cemented between Mr. Hewitt and Mr. Cooper a lifelong friendship, subsequently confirmed by

business association, and by the marriage of Mr. Hewitt with the sister of his friend, Peter Cooper's only daughter.

Upon his return from Europe, Mr. Hewitt completed his legal studies, and was admitted to the bar in 1845. But he soon relinquished the practice of the law, and formed with Edward Cooper the firm of Cooper & Hewitt, to assume that branch of the business of Peter Cooper which related to the iron-manufacture.

I pause here, to note the singular combination of influences and events which prepared Mr. Hewitt for his subsequent career of peculiar usefulness and honor, as a patriot, statesman, publicist and commercial pioneer. His ancestry, on both sides, was such as to bequeath to him sturdy independence, self-reliance, refined tastes, conscientious motives, generous impulses, and that love of liberty and faith in democracy which had made his parents citizens of the American Republic. Moreover, it is fair to say that the father's connection with the first American steam-engine and the connection of that machine with American mining were more than accidental omens of future intimate and influential connection of the son with that marvelous national progress, of which the mineral wealth of the United States has been the primary basis and steam-machinery the indispensable agent.\*

Again, Mr. Hewitt's study of the law gave him a grasp of principles, and an insight into their operation indispensable to his equipment, both as a business man and as a statesman.

And, finally, his relations with Peter Cooper must have lent additional hope and enthusiasm to his patriotic ambition.†

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It was fortunate for Peter Cooper that his son and his son-in-law took charge of his affairs when they did. For he was a business man of the old school, accustomed to superintend all details himself, and very largely to keep the particulars of his financial situation "in his head." His industry, honesty, frugality, and inexhaustible inventive genius had built up a busi-

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\* See Mr. Hewitt's Presidential Address, already cited.

† In my "Life of Peter Cooper," I have set forth the character of that great and good man as an incarnation of the American courage, faith and hope of the first age of our Republic.



ness greater than his primitive methods could handle. But he wisely acquiesced in the changes required by new commercial conditions, and lived to see them amply justified by results.

One of these changes was the removal to Trenton, N. J., of a wire-mill owned by Mr. Cooper, and occupying in New York City a site no longer suitable. In its new location, this mill became the nucleus of the large business subsequently developed by Cooper & Hewitt, and represented by the works of the Trenton Iron Company and the New Jersey Steel and Iron Company, at Trenton, and the iron blast-furnaces at Phillipsburg, Ringwood and Pequest, N. J., and Durham, Pa. For many years the firm was a leading representative of the iron-manufacture of the United States. It was the first American concern to make iron beams and girders, and the third to roll iron rails. Its blast-furnaces at Phillipsburg and Durham were regarded, in their day, as models of scientific design.

Foreign as well as domestic improvements were recognized and promoted by this house. In 1856, within sixty days after the reading, in England, of Bessemer's famous paper, an experimental Bessemer converter was running in the works of Cooper & Hewitt, at Phillipsburg.\* This experiment, it is true, had no immediate practical result of value. In fact, it must be said to have failed, as did similar first experiments elsewhere. Yet it deserves mention as evidence of the alert intelligence with which Mr. Hewitt noted and appreciated every sign of progress in the metallurgy of iron and steel, at home and abroad.

But the next important illustration of this characteristic quality did not end in failure. Mr. Hewitt, as United States Commissioner to the Paris Exposition, in 1867, made a report upon the iron-manufacture and trade of the world, as there exhibited, and the bearing of that exhibition upon the actual and prospective iron and steel industry of this country, which took at once the position it still holds as a classic in the literature of its theme. It was, indeed, only the first of a series of equally noteworthy (though, perhaps, less widely famous) utterances of criticism and prophecy which history has already verified. But he did not stop with words. Recognizing the

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\* See the Address of Sir James Kitson, accompanying the presentation to Mr. Hewitt of the Bessemer medal, in 1890. *Trans.*, xix., 516.

value of the new open-hearth method of steel-manufacture, as shown at Paris by the Martins, he undertook its introduction into this country, acquiring control of the Martin patent, erecting at Trenton the first American open-hearth furnace, and urging upon manufacturers in every possible way the merits of the new method, then questioned, but now universally acknowledged.

Still later, Mr. Hewitt interested himself in the invention of Mr. G. J. Snelus for the use of basic linings, which, combined with the subsequent improvements of Thomas, became the foundation of the "basic" processes, now so widely and successfully used. It was for these and other services to the processes of the iron- and steel-manufacture that the Iron and Steel Institute conferred the Bessemer Gold Medal upon Mr. Hewitt in 1890.

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Turning from this inadequate sketch of one department only of Mr. Hewitt's long activity, I must make mention—even less fully and fairly, I fear—of his political career. Though always an active and eminent member of the Democratic party, he gave to the government, during the War of the Rebellion, a hearty and effective support. In fact, the works of Cooper & Hewitt were for four years practically devoted to the public service in the manufacture of gun-barrel metal and other munitions of war, without profit to the owners, but with great advantage to the Union cause, as the War Department repeatedly acknowledged. Perhaps the most striking instance was the rapid construction, in 1863, by a *tour de force*, of the mortar-beds for General Grant's expedition against Fort Donelson, and for the fleet of gunboats which, under Commodore Foote, subsequently opened the Mississippi river. It is believed that the prompt supply of these mortar-beds saved from failure both of the military operations mentioned.\*

In 1875, Mr. Hewitt held office for the first time, as a Representative from New York in the Forty-fourth Congress. This position he retained, by successive re-elections, for twelve years, resigning it, in 1887, to become Mayor of New York. His career in Congress was both eventful and useful in the highest degree. Differing freely and often with the prevailing tenden-

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\* In this remarkable performance, other iron-works co-operated, but the chief credit belongs to Cooper and Hewitt.

cies of his party, he was, nevertheless, its guide in many matters, and was able to shape much wise legislation. Among the objects of his Congressional work, the following may be mentioned :\*

1. The improvement of the consular and diplomatic service.
2. The resumption of specie-payments (a repeal of which step he successfully opposed).

3. The overthrow of the "free-silver" heresy, which he sternly opposed, even making, in 1876, an exhaustive argument to show the dangers of the then impending Sherman compromise, authorizing "limited" silver coinage, which, in fact, deranged for twenty years the business of the country and increased the final difficulty of a return to sound financial principles.

4. The unification of the various geological surveys of the United States, and the creation of the National Geological Survey, which now, in a systematic manner, covers the whole of this field. It is fair to say that Mr. Hewitt's remarkable speech on this subject conclusively settled the question at issue, and won the day for the proposed new national enterprise.

5. The advocacy of liberal revenue-legislation, and especially of free raw materials for manufacture. Mr. Hewitt's course in this respect was naturally opposed by the extreme Republican protectionists, while, naturally also, it alienated the extreme free-traders of his own party. But he was really following Hamilton's theory of protection, which he had always held, and which, in its essential principles, recognizes the wisdom of low duties (or even no duties), and of commercial reciprocity, apart from all general formulas, according to the conditions of each period and each concrete problem of industry and trade. Mr. Hewitt's speech of 1886, on the Mexican treaty, was a clear utterance in favor of the reciprocity then opposed by Blaine, McKinley and Reed, but subsequently favored by all of them, and now asserted as the Republican policy.

6. Outside of all general theories or party-platforms, Mr. Hewitt's thorough knowledge of business, as well as finance, was valued by both sides, and freely placed at the service of

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\* This list, though incomplete, possesses peculiar interest, as comprising the items which Mr. Hewitt himself, many years later, named to me as those which he recalled most clearly, and considered most important.

both. The administrative reforms successively embodied in the Morrison, Mills, McKinley and Dingley Tariff Acts were his work; and Mr. McKinley, in his famous speech in advocacy of his own bill, frankly acknowledged the services of Mr. Hewitt in this particular.

7. In 1878, he served as Chairman of the Congressional committee on the grievances of labor; and his shrewd, kindly colloquies with the partisans, theorists and cranks who flocked to give "evidence" before that body, conferred upon its voluminous records an interest and value which they would not otherwise possess.

8. As a representative of the metropolis, he secured the appropriations necessary for the deepening of the ship-channels, and induced the Federal government to commit itself to the systematic and extensive improvement of New York harbor, which has been steadily prosecuted ever since.

9. In 1876, Mr. Hewitt, as chairman of the National Democratic Committee, had charge of the Tilden campaign, in which the "literary bureau," organized by him, received general commendation. He was a member of the special Congressional committee which framed the bill for an Electoral Commission to settle the disputed result of that contest; and, in the House of Representatives, he led his party in accepting and executing the provisions of that bill.

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In 1887, Mr. Hewitt resigned his Congressional position to become Mayor of New York, after a memorable contest in which the late Henry George was the opposing candidate. His work as mayor consisted chiefly in restoring efficiency to the city government and in securing the proper revenue from the municipal resources and property. Tammany Hall, which had supported his candidacy as a choice between two evils, soon developed an active opposition to his absolutely non-partisan administration, and was able, under the legal limitations then surrounding the Mayor's power, to thwart many of his plans and measures. Nevertheless, at the close of his term, he left the machinery of all the departments in a satisfactory condition of efficiency and ready for further enlargements of scope and power. And, besides this administrative improvement, he

managed to impress upon his fellow-citizens worthy ideas of future metropolitan development, such as ordinary reformers, intent on stopping leaks and abuses, are often afraid to propound. His novel practice of issuing, not merely one annual message, covering with vague generalities more topics than the public could at one time digest, but a series of important messages, each devoted to a single subject, and presenting that subject so forcibly as to command popular attention, contributed greatly to the political education of the whole community.

A striking illustration is the plan for rapid transit, which he formulated in one of these notable messages. It was opposed by Tammany Hall, but subsequently, under the pressure of public opinion, was adopted by the Legislature; and it is now, with the support of all classes in the community, in process of execution. Its main feature is that the work, constructed with the money, and being actually the property, of the city, shall be leased to a responsible corporation, at a rental covering both the interest, and a sinking-fund which will in fifty years repay the principal, of its cost; so that, at the end of that period, the whole rapid-transit system will be the property of the city, free of all cost, even in the way of taxes *ad interim*, and equally free from all obligations to any private or corporate interest. A fairer, wiser, or more ingenious solution of the difficult problem of disposing of a great public franchise, it would be difficult to imagine. If something of this kind had been done with regard to the elevated and surface railroads of New York, how great would be the gratitude of the next generation! The Chamber of Commerce, recognizing the great public service of Mr. Hewitt in this respect, presented to him a gold medal in commemoration of it; and probably this will be regarded hereafter as the most important achievement of his public career, judged, as men are prone to judge, by results, rather than by arduous labors or heroic sacrifices.\*

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\* In April, 1900, the New York Chamber of Commerce, of which Mr. Hewitt had been a member since April, 1861, elected him to honorary membership, "in consideration of his long and valuable services to the city, State and nation, and with special regard to his initiation in this body of the rapid-transit plan, under which the contract was awarded and the work is now proceeding." At the same meeting, the gold medal mentioned in the text was ordered to be prepared "in recognition of the eminent services of Hon. Abram S. Hewitt in the cause of civic rapid transit under municipal ownership, and that it be presented to him

But that is not the only, or the highest, standard of measurement; and, to my mind, the most meritorious (and, in the long run, the most fruitful) of Mr. Hewitt's numerous activities was his patient, wise and incessant administration, for more than forty years, of the affairs of the Cooper Union, of which he was from the beginning, as secretary of the board of trustees, the executive manager. The credit for this great institution—a pioneer of its kind—has been justly given to Peter Cooper; but he would have been the first to acknowledge the essential service rendered by those to whom he entrusted his work, and who shaped, organized and administered it, with his approval, so long as he lived, yet not at all in accordance

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by the President, with assurances of the admiration, respect and affectionate regard of his fellow-members of the Chamber of Commerce."

On that occasion, Mr. A. E. Orr, chairman of the Rapid-Transit Commission, said:

"In 1888, Abram S. Hewitt, then Mayor of New York, in his annual message, called attention to the pressing need of real rapid transit, and earnestly advocated municipal construction as the only means of obtaining it effectively. The Board of Aldermen gave little heed to his suggestion. . . . Nothing daunted, however, Mr. Hewitt prepared a bill illustrative of his views, and had it introduced at Albany. It was referred to the Committee on Cities, and there it was smothered."

Speaking next of the action of the Chamber of Commerce, in 1894, in appointing a committee to investigate and report upon the rapid-transit situation, Mr. Orr said:

"It was then that the transcendent genius and foresight of Mr. Hewitt, and the benefits of the previous study he had given to the whole question, came into full play."

After sketching the main features of the law creating the Municipal Rapid-Transit Commission, the speaker continued as follows:

"The entire credit of inventing those provisions belongs to Mr. Hewitt, and the more critically they are examined, the more remarkably advantageous to all parties in interest they appear. . . . Mr. Hewitt, because of increasing years and impaired health, declined to serve upon the Commission, but it has always received his warm sympathy and the benefit of his wise counsel. He also served on the Board of Experts appointed by the Commission to pass upon the methods of construction, plans, specifications and estimates of its chief engineer, giving to this work many weeks of valuable time, but refusing to receive the honorarium awarded to each member of the Board for that service. . . . Mr. Hewitt's long and honorable life has always been prominent, whether in the halls of Congress, in our civic government or in private enterprise. The records of this Chamber sparkle throughout with his brilliant speeches and reports in furtherance of the best interests of this country and its commercial metropolis; but I cannot help believing that his splendid service in the cause of municipal rapid transit is the crowning act of his life, and will perpetuate his memory and render New York his debtor as long as the city shall exist."

with the crude features of his original dream of it. For particulars, I may refer to my "Life of Peter Cooper."\* Let it suffice to say here, from personal knowledge, that while the inspiration of this great enterprise came from the heart of Peter Cooper, the wise and effective execution of it was largely due to the ability and devotion of Abram S. Hewitt. At various times, and especially in January, 1902, he made (jointly with his family and those of Edward Cooper and other relatives) liberal additions to the permanent endowment of the Cooper Union. But these large gifts of money were not so praiseworthy, in my judgment, as his life-long, patient toil in the work of the Cooper Union itself.

I need hardly add that Mr. Hewitt was identified with many public movements for education or charity, and that his utterances on these subjects were marked by a rare combination of sagacity with sympathy.

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He became a member of the new American Institute of Mining Engineers in 1871, and was twice (in 1876 and 1890) elected its President,—accepting this office, on both occasions, at considerable personal sacrifice, because special conditions indicated him as the one man for the place. We had in our membership great engineers, like Holley, and many capitalists of larger wealth than Mr. Hewitt; but we had no one who could so fully represent the country and the Institute to other nations. In 1876, it was the Centennial Exposition, and, in 1890, the visit of the Iron and Steel Institute and of the German Ironmasters' Association, which called for such an official representative. On both occasions, Mr. Hewitt not only assumed the nominal dignity of President, but contributed to the proceedings addresses of permanent value, prepared in the midst of other duties, engrossing and exhausting.

The list of his formal contributions to the *Transactions* is as follows:

"A Century of Mining and Metallurgy in the United States" (Presidential Address at the Philadelphia meeting, June, 1876), *Trans.*, v., 164.

"Iron and Labor" (Presidential Address at the New York meeting, September, 1890), *Trans.*, xix., 475.

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\* Houghton, Mifflin & Co., New York, 1901.

“Reply to the Address of Sir James Kitson” (accompanying the presentation of the Bessemer medal, September, 1890), *Trans.*, xix., 517.

Each of these utterances was profoundly important, attracted wide attention, and was republished and quoted frequently on both sides of the Atlantic. But earlier, and in some respects more significant than any of them, was the address of welcome delivered by Mr. Hewitt at the New York meeting of May, 1872, which might be entitled “Iron and Civilization,” if it did not cover also a larger theme.

That address, I regret to say, is not to be found in the volumes of *Transactions*. The reason of its omission (which seemed conclusive then, as it seems almost ludicrous now) was, that the Institute, comprising but a couple of hundred members, published no volume during the first two years of its existence. In the third year, the first volume of the *Transactions*, comprising 475 pages, and containing, as the preface explained, “an abstract of the proceedings of the first two years, and a number of papers, selected from those proceedings,” was issued, on the personal pecuniary responsibility of the Council—a responsibility, I hasten to add, which the rapid growth of the Institute soon reduced to zero. Under the circumstances, some of the less important papers, and all the welcoming addresses, not formally classed as papers, were omitted from Vol. I., in order to reduce its cost; and, under the application of this rule, the address of Mr. Hewitt, to which I here refer, shared the fate of those of merely temporary and ceremonial nature. Later history has emphasized its prophetic character; and I feel justified in reprinting it now, as an appendix to this sketch.

Mr. Hewitt was one of the earliest honorary members of the Iron and Steel Institute, and held a similar relation to many other scientific associations. In 1887, he received from Columbia the degree of LL.D., and he was a member of the Board of Trustees of that university and Chairman of the Trustees of Barnard College, as well as Trustee of the Carnegie Institution, the Burke Foundation, the Manhattan Eye and Ear Hospital, and many other public charities.

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His latest public utterances were not less wise and vigorous



than those of earlier years. I may mention, for instance, his argument in October, 1902, before the Rapid-Transit Board, in favor of the Pennsylvania Railroad Tunnel franchise. Again, his characteristically frank, acute and sensible utterances (including his open letter to Mr. John Mitchell) concerning the anthracite coal-strike of 1902 are classic already, and will long furnish instruction and inspiration to the lovers of liberty and justice, as incarnated in American institutions. And who can forget the ringing letter sent by him to the John Fritz anniversary-dinner, October 31, 1902, the reading of which brought to their feet, in irrepressible enthusiasm of applause, the "captains of industry" constituting that great assembly? Indeed, that letter may fitly form a part of this present narrative of a life, the spirit of which it so well epitomizes. Here it is:

"As you are aware, I have been debarred for some time by the limitations of age from assisting at any function which takes place in the evening. I have been trying, however, to make an exception in favor of the complimentary dinner given by his admirers to my old and valued friend, John Fritz; but, now that the time has arrived, I find that I must deny myself the pleasure of personally congratulating him, in the company of his loving friends, upon the attainment of his eightieth birthday in the full possession of his health and faculties, and with the promise of many honorable years to come.

"We have made the journey of life together, and, to some extent, along the same lines of action. It is pleasant to recall that, during the half-century of our association, at times of competitive struggle, the friendship which has existed between us has never in the slightest degree been disturbed. This happy experience is due, doubtless, to the amiable traits of Mr. Fritz's nature, which, with all its masculine energy, is tempered with the sweetness of the gentler sex. 'Once a friend, always a friend,' will be inscribed upon the record of a career which, in some respects, is unique among the men of our day and generation.

"I do not intend to indulge myself in recounting any of the interesting details of his long and useful life. This pleasant duty will be performed by others, but I do desire to point out that the life-work of John Fritz affords a very conspicuous example of the working of American institutions during the century which has just closed, the most remarkable era of progress in the history of the human race. That a boy, born in humble life, with no advantages of education or opportunities for position, without influential friends, or the favoring accidents of fortune, should be able to advance steadily in usefulness, power and the respect of his fellow-men until, by common consent, he occupies the first place in the domain of the practical industry with which he has been connected, gives conclusive evidence that political institutions which afford free play to individual ambition, industry, ability and strict integrity are worthy of all loyalty, and should be cherished and preserved at all costs and hazards.

"The developments of the twentieth century show that these institutions are in great peril. Their essence is to be found in individual liberty, involving the rights of free labor and the acquisition of private property under lawful condi-

tions. When the right of free action shall be suppressed, the possibility of a career like that of John Fritz will be destroyed. Collectivism, ending in socialism, may afford other advantages, but let it not be overlooked that those advantages will be obtained only by the sacrifice of personal freedom, and will arrest the progress of civilization due, during the ages that have passed, to the substitution of freedom for force.

"John Fritz is a living proof of the results of individual and industrial liberty in a country endowed with boundless resources. In vain shall we seek for a like career in nations or in countries where the individual initiative has been suppressed. The stagnation of China, whose men are physically strong, and whose resources are abundant, is in marked contrast with our own land, where, heretofore, every citizen has been free to employ his labor and his energies in his own way, so long as the rights of others are respected.

"John Fritz, therefore, is to us more than a man whom we love and respect, more than a friend to whom we wish many years of health and happiness; he is an example of the free spirit of American institutions,—a beacon-light, warning the present and coming generations against permitting any invasion of the principle of the liberty of the citizen, which alone has made our beloved country great and free."

Free opportunity; full justice; fair play; no coddling by government; no tyranny of caste; no sacrifice of liberty; no maudlin sympathy with the disturbers of order and the violators of law; no corrupt fear or favor, either towards those who possess capital or towards those who covet it; no power without responsibility; no premium upon clamorous idleness and no blackmail to the "sturdy beggar"—this was the wholesome, manly, old-fashioned, American creed of Abram S. Hewitt. On the other hand, his sense of duty to his city, his nation and mankind knew no limits save those of his strength and ability. "Millions for help, though not a cent for tribute!" might have been his motto. May it be long before such an ideal of blended justice and generosity shall cease to be followed among us!

Revisiting Rome after an absence of forty-three years, I have been impressed with the mighty changes wrought during that period. A splendid new metropolis is growing up around and amid the ruins of the old. To me, resurrection is more wonderful than death; and I confess that I feel, in the presence of the fresh, strong life of young Italy, a thrill which all the remains of its mingled past of heroism, conquest and decay do not excite. This spirit is wonderfully expressed in the magnificent Garibaldi monument, which now stands upon the Janiculum, commanding a more magnificent panorama of Rome than can be seen from any other of the many heights in or

near the city. Upon the splendid architectural pedestal, with its subordinate groups of sculptured allegory and history, stands the colossal equestrian statue—both steed and rider in vigilant repose, and the hero looking keenly over the vast expanse of old, new, eternal Rome to the horizon of the Sabine and the Alban hills. As my coachman, an ardent Roman patriot, tersely said, “Ecco! *Our* Garibaldi, watching over *his* Rome!” There is, in bronze and stone, combined with site and story and significance, nothing grander in the world.

Not as seated in the soldier’s saddle, surrounded by the emblems of war, would we represent our great civic hero. Yet, remembering his patient labors, generous sacrifices, wise, far-reaching plans and splendid achievements for us, we may well think of him now as, from some superior height, looking—*our* great citizen—upon *his* beloved city!

ROME, January 19, 1903.

#### APPENDIX.

The following is the address of Hon. Abram S. Hewitt, at the New York meeting of the Institute, May, 1872, as published in the *Engineering and Mining Journal*, May 28, 1872:

“*Mr. President and Fellow-Members of the American Institute of Mining Engineers:* To me is assigned the honor of bidding you welcome to the city of New York, on this, the occasion of our first annual meeting, and I am sure that you will find yourselves made welcome by all who have the honor and prosperity of the city at heart. For New York, although far from being the cherished home of science and art, comprehends that its growth and its future greatness depend upon the development of the natural resources of the country of which it is the commercial metropolis; and it is sufficiently enlightened to understand the necessity of scientific knowledge and trained experience for the attainment of the most useful and profitable results from industrial enterprises. No body of men can understand better than you that capital is essential to the development of natural resources on the scale demanded by modern civilization; but capital does not always comprehend as fully that science and experience are essential for the profitable use of money in the vast industrial undertakings of our day, and hence result great waste of resources and disastrous failures. A few considerations may serve to shed some light on this subject, of such material consequence to science and capital, and, at the risk of overstepping the conventional limits of a formal welcome, I venture briefly to suggest them as the means of establishing a common ground of sympathy and fellowship between the men of science and the men of business, and I am sure that you will pardon me if I draw my illustrations from that subject with which I am most familiar—the production of iron. In 1856, I had occasion to trace the history of the manufacture of iron, and established what may be termed its law of development, rather rude, indeed, but plainly dependent upon the growth of population and the spread of civilization throughout the world.

"At that time, the annual production of iron had reached about 7,000,000 tons, of which Great Britain produced 3,500,000 tons, and the United States about 1,000,000 tons. The consumption of Great Britain was 144 pounds, and of the United States 84 pounds, while the average consumption of the world was only 17 pounds per head of population. It was shown that the consumption per head was steadily on the increase, and that, consequently, the annual production was being enlarged so rapidly as to double once in 14 years; and it was predicted, after making due allowances for all drawbacks, such as the wars which have, unhappily, taken place in the interval beyond any possible expectation, that in 1875 the production of iron would surely reach 14,000,000 tons. The actual returns show that, in 1871, the production amounted to 13,500,000 tons, and, in 1872, the limit, 14,000,000, will, undoubtedly, be passed, so that the estimate made in 1856 is more than realized. Meanwhile, the consumption has risen in England to 200 pounds, in the United States to 150 pounds, and in the whole world to 30 pounds per head. It is not possible to convey a more striking idea of the progress of the world, during the last 17 years, than this statement affords. The consumption of iron measures the progress of civilization, and it is impossible not to believe that the whole world will, ultimately, require as much iron per head as we now use in the United States, when a total annual production of over 70,000,000 tons will be required. But if these figures seem to be at all wild, no one can for a moment doubt that the next 17 years will double the present annual production of iron, bringing it up to 28,000,000 tons per annum; and I feel quite safe in asserting that the beginning of the 20th century, which some among you may hope to see, will witness an annual production of over 40,000,000 tons.

"You need not be told that iron is produced at less money cost in Great Britain than in any other quarter of the globe. This has enabled her to produce about one-half of the total annual make. Of the 7,000,000 tons made in 1855, Great Britain produced 3,585,906 tons, and of the 13,500,000 tons produced last year, she turned out nearly 7,000,000 tons. It is evident, however, that there are limits in the way of raw material and labor beyond which Great Britain cannot go. While I see no reason to doubt that there will be a steady increase in production, it is evident that she will not be able to supply hereafter, as heretofore, so much as half the annual wants of the world of iron. But, allowing this proportion to Great Britain, there will still remain 14,000,000 tons to be made by the rest of the world. The history of the trade, as well as the natural resources of the several nationalities, prove that the bulk of this additional product can only be made in the United States. We are, in fact, the only people who have kept pace with Great Britain in the ratio of increase. In 1855, when Great Britain produced 3,500,000 tons, we produced 1,000,000 tons. In 1872, when Great Britain will produce 7,000,000 tons, we produce 2,000,000 tons—the quantity produced in Great Britain in 1847, showing that we are only 25 years in arrear of her magnificent production. At the same rate, therefore, we would make 7,000,000 tons in 1897. But as Great Britain cannot possibly maintain her rate of increase, there does not seem room for a doubt that our annual production will reach, at least, 10,000,000, and will, probably, amount to 15,000,000 tons before the close of the present century. This means that 25,000,000 to 40,000,000 tons of iron-ore shall be annually extracted from our mines, and that our coal-production will exceed 100,000,000 tons per annum, required for iron and other branches of industry. It means that an investment of capital to the amount of \$500,000,000, at least, and, probably, \$1,000,000,000, shall be made in opening mines, erecting works and supplying the requisite machinery of production. New York is already the financial center of the American continent, and is destined to be the

main distributor of capital for the world. This vast sum of money will, therefore, be drawn from the accumulations of capital controlled in New York, and its productive results will depend mainly upon the judgment and skill displayed in its expenditure. Here, then, is the common ground on which capital and science must meet and shake hands and be, henceforth, inseparable friends.

"But if it be the mission of science thus to reconcile capital with industry, it is the still higher and nobler mission of science to reconcile industry with capital. The world is full of the conflict between capital and labor. When there should be peace, there is war. When nature intended an absolute harmony, there is utter discord. For one, I am free to say, after the most careful investigation, and very extensive observation, that iron has heretofore been made at too low a cost in foreign countries to allow the workmen engaged in its production a fair share of the necessities and comforts of life. This is due to the fact that the possession of virgin resources in coal and iron made it easy to increase production beyond the present wants of society. The resulting competition has had the effect to reduce prices to so low a point that proper wages could not be paid, and mankind has been enabled to get cheap goods at the expense of humanity itself.

"I thank God, reverently and with gratitude unspeakable, that this day has passed, I trust, forever. These virgin resources are mainly exhausted, and it is no longer possible, in Europe, at least, to produce more iron than the world requires; prices have risen; the workmen are demanding and receiving a more reasonable reward for their labor, and it now only remains for them to fit themselves and their families for a rational use and enjoyment of the fruits of their toil. In looking back over the sad and gloomy fields of suffering among the European mines and works, which I have traversed so often, and in looking forward to the more cheerful prospect now spread out before the sons of toil, I am tempted to exclaim with the Patriarch: 'Lord, now lettest thou thy servant depart in peace, for mine eyes have seen thy salvation.'

"But cheap iron is a blessing to mankind, and to deprive the world of it is a calamity so serious that no one can contemplate it without a feeling of reluctance. Here, again, science steps in to reconcile high wages with cheap iron. It is the mission of science to cheapen processes, which enables wages to be raised without enhancing the cost of the product to the world. The history of industry is full of examples of the truth of this proposition, but for our purpose the Bessemer process affords its best illustration. By the genius of one man the whole world is enriched, its comforts enlarged, its progress promoted, and new fields of art and industry opened to its enterprise and energy. The annual saving in carrying on the business and transportation of the world can only be measured by millions; and when equal genius is applied to the proper distribution of the savings produced by the Bessemer process, by the Danks Puddler, and other economical processes that have been and will be invented, the laboring classes all over the world will be lifted out of the depths, and this earth become the paradise it was intended to be, when the Great Giver of all endowed it with so much beauty and such boundless sources of wealth, and made the forces of nature to be the servants of man, whenever he learns how to use and govern them. You, gentlemen, have limited yourselves to the study of physical laws and their application to industry; but I hope to see the day when all over this land, and throughout the world, there will be similar associations devoting themselves with equal zeal and intelligence to the discovery of the laws upon which society should be organized, and to the application of these laws to the proper distribution of the fruits of industry among those who labor for their production; so that nowhere in the world, and least of all in this land of boundless resources, shall it be said that there are idle

hands because there is no work to be done, or that there are want and misery because there is not a just division of the proceeds of industry.

"If, then, my views in regard to the dignity and importance of your mission be correct, you have not associated yourselves together one day too soon. You can derive encouragement from the magnificent results achieved by your sister association, the British Iron and Steel Institute, which, though only two years your senior, has already given to the world several volumes of inestimable value, and among them that admirable treatise of I. Lowthian Bell, on 'The Chemical Phenomena of Iron-Smelting,' wherein the laws governing the operation of the blast-furnace are placed upon settled foundations, and two continents have been made his debtor, a debt which you will gladly join with me in recognizing on the first suitable public occasion which has occurred since the completion of his great work.

"Having thus briefly traced out the mission of science in our day to bring capital into productive relations with labor, and to remove the just grievances of labor, not against capital, but against its ignorant administration, and to make commodities cheap for the benefit and not at the expense of humanity, let me, in conclusion, sketch the picture which will be presented at the beginning of the next century, when our mining interests will be developed on a scale somewhat commensurate with the area of the country and the extent of its resources.

"As New York will be the center of capital, so will it be the initial point of our iron and steel industry. On the shores of the Hudson river, the ores of Lake Champlain, of the valleys of Connecticut, and of the highland ranges of New York and New Jersey will meet the anthracite coals of Pennsylvania upon conditions so favorable that New York and its vicinity must become a great metallurgical center. Thence the chain of fire, extending across New Jersey and following the banks of the Lehigh and Schuylkill to the Susquehanna, will lead us by the margin of the coal-fields, along the outcrop of the magnetic, hematite and fossiliferous ores which extend through Pennsylvania, Maryland and Virginia, North and South Carolina, Tennessee, Georgia and Alabama, nearly to the Gulf, so that the light of furnace answering to furnace will never be lost sight of in the long line of over 1000 miles. Thence, turning to the West, Missouri, Kentucky, Western Tennessee, Ohio, Indiana and Illinois will be all aglow with furnaces, forges and mills fed by the admirable fuel of the inexhaustible coal-fields of the West and the superb ores of Missouri and Lake Superior. The waters of the great lakes will reflect the flames which will light up their margin, while to the west, along the line of the various Pacific railways, the newly-found coal and iron of that hitherto trackless region will form an enduring basis for the growth of industrious communities, busy cities and teeming farms. The West Coast will not be behind in the race, but an iron industry, more valuable than its mines of gold and silver, will yet supply its growing millions with the fundamental basis upon which modern civilization rests. The growth of this vast industry will be accompanied by the schoolmaster, the preacher and the physician. Homes, of which human nature may be proud, will be established in its wake, labor and Christianity will march hand in hand, binding all interests and all classes so harmoniously and so indissolubly together that peace and good-will between capital and labor shall prevail throughout the land forever.

"Gentlemen, you are welcome to New York."

The prophecies made in the foregoing address as to the future production of pig-iron were strikingly fulfilled. In 1872, Mr. Hewitt estimated that the world's annual product

would be, after 17 years, 28,000,000 tons—it was 27,630,000 tons in 1890; that it would be, at the beginning of the 20th century, over 40,000,000 tons—the average of 1900 and 1901 was 40,499,786 tons; that in 1897 the United States would produce 10,000,000 tons—the actual product of that year was 9,652,680 tons; and that the annual product of the United States at the beginning of this century would be 15,000,000 tons—it was, in fact, 14,960,000 tons (the average of 1900 and 1901). On the other hand, his estimate as to the United States product of iron-ore was too large, because he could not foresee the enormous development of very rich iron-ores in the Lake Superior region, which furnished, in a smaller tonnage of raw material, the iron required for our furnaces; and his estimate of United States coal-production was too small, because he could not foresee the immense demand for other branches of manufacture, transportation, etc.

From the reply which, as President of the Institute, I made on that occasion to Mr. Hewitt's address, I quote the following paragraph:

“This is no occasion for fulsome flattery; but it is an occasion for the honest recognition of the fact, that the true glory of New York must be sought, not in the concentration of wealth and power, but in the uses of that wealth and power, and their influence upon civilization, education and manhood. This stately building, beneath the roof of which we meet to-night, is a monument of such a glory; it is Peter Cooper's witness to the value of scientific education for workingmen, and the duty of society towards its own members. I need not add, for the fact is too well known, that if the patience, self-sacrifice and wise benevolence of Mr. Cooper established this institution, its successful organization and its vigorous administration through many years have been largely the work of Mr. Hewitt. I trust that many of you will find occasion to inspect the Cooper Union throughout while you are in the city. There is not its counterpart in the world. Not in London, not in Paris, not in Berlin, is the poor man or the poor man's son, already perhaps under the necessity of daily labor for daily bread, so provided with the free means of self-improvement and advancement. Examine these well-appointed laboratories and lecture-rooms, yonder library and vast reading-room, now crowded with eager students; consider the steady stream—nay, the meridian glow—of influence which proceeds perpetually through books and journals and lectures and instruction from this blazing center; remember that a citizen of New York spent forty years of his life and half his fortune to build up this vast agency of benefaction as a repayment of the debt which, in his opinion, he owed to his poorer fellow-citizens, and that another has devoted time and strength, without counting the cost, to the arduous work of developing and maintaining it; and I know you will say with me that the Cooper Union is the glory of New York, and that no man has a better right to speak for New York than he who represents this greatest element in her metropolitan grandeur—the free diffusion of knowledge among the people.”

## Effect of Fineness of Grain on the Fusibility of Clay.

BY HEINRICH RIES, ITHACA, N. Y.

(Albany Meeting, February, 1903.)

THE fusion-point of clay has usually been considered to be influenced chiefly by the quantity of fluxing impurities which the clay contained. Those clays which contain a high percentage of fluxes, such as ferric oxide, lime, magnesia, and alkalies, fuse at a low temperature.

Aside from the percentage of fluxing-impurities, the size of the grains exerts an important influence. Yet, while this fact has, no doubt, been recognized by many, practically no attempts have been made to prove it by actual experiment, although in the case of fire-clays, Prof. H. O. Hofman has rather disputed it.

To illustrate the point more clearly, suppose a clay to be made up of a mixture of very refractory grains, such as kaolinite, and some fusible ones. If this clay be heated, the more easily fusible grains will melt first on the surface, and the softening under heat will gradually extend from the surface of the grain to its center. The larger, therefore, the size of the grain, the longer the time required for the heat to penetrate it. Now, if two samples of clay contain the same volume of flux, but of unequal size, the one containing the smaller-sized grains should fuse first, because the fluxing materials are more evenly and thoroughly distributed through the clay, so that they will more or less surround the refractory grains, which tend to act as a framework and hold up the clay-mass when heated.

In order to demonstrate the effect of fineness of grain on the fusibility of clay, several mixtures were made up, consisting of white clay and other minerals. The white clay used was a very refractory type from Georgia, the fusion-point of which lay about cone 35 of the Seger series. The mixtures were formed into small bars  $1\frac{1}{4}$  inches long, and set on a narrow surface, so that the ends were free to bend as soon as the mass softened. They were burned in a Seger gas-furnace.



One set of mixtures consisted of the following:

A. Equal parts of clay and hornblende, the latter being ground to pass through 150-mesh.

B. The same as A, but the hornblende passed through a 100-mesh screen and collected on 150.

C. The same as A, but the hornblende collected on 100-mesh screen.

When burned to Cone 5, the three were little affected, except that the one with the finest grains was colored uniformly red, while that with the coarsest presented a speckled appearance. When burned to Cone 8, the bar of mixture A was considerably bent at both ends, while that of B was nearly straight, and C was perfectly straight. At Cone 10, B was thoroughly fused, and C slightly bent.

This seemed to show well the effect of grain-size in the case of hornblende. The object of taking such a large amount of fluxing-material was simply to get results at moderate temperatures.

A second set of mixtures was then made up, consisting of equal parts of clay and calcite. The results obtained were similar to those found in the case of hornblende, but occurred at higher temperatures, since the calcite is a less powerful flux.

The mixtures were as follows:

D. Equal parts of clay and calcite, the latter ground to pass through 150-mesh.

E. The same as D; calcite passed through 100-mesh and collected on 150-mesh.

F. The same as D; calcite collected on 80-mesh.

These three, when burned to Cone 10, gave the following:

D. Bar fused, and bent, but not viscous.

E. Bar well vitrified, slightly bent, and edges sharp.

F. Bar straight, with but little evidence of fusion.

It appears quite clearly, therefore, that the size of grain exerts a marked influence on the point of fusion of the clay, and it would seem reasonable also to look for similar effects from other minerals. The writer has a series of experiments under way, the results of which will be presented later.

## Geological Relations of the Manganese Ore-Deposits of Georgia.\*

BY THOMAS L. WATSON, GRANVILLE, OHIO.

(Albany Meeting, February, 1903.)

### INTRODUCTION.

A PART of the field-seasons of 1900, 1901 and 1902 was devoted by the writer to a study of the manganese ore-deposits of Georgia. A report embodying the results of this study is rapidly nearing completion, and when finished it will be published as a bulletin of the Geological Survey of Georgia. I believe, however, the importance of these deposits warrants a separate and earlier publication than that made possible by the Survey, in which the more important results of the study are briefly set forth. It is principally for this reason that the present paper has been prepared.

*Previous Work.*—Some of the more important deposits of manganese occurring in the Paleozoic area of Georgia were studied to some extent previous to 1890 by Dr. Penrose;† and again in the early nineties by former State Geologist, J. W. Spencer.‡ In a recent paper published in the *Transactions*, on the "Geological Relations of the Iron-Ores in the Cartersville District, Georgia," Dr. Hayes§ briefly mentions the manganese-ores of the same district.

*Distribution of the Manganese-Ores.*—Georgia is divisible into three geologically-distinct areas which, named in order from southeast to northwest, are (a), the Coastal Plain; (b), the Crys-

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\* Published by permission of the State Geologist of Georgia.

I wish to make acknowledgements to the various excellent maps and publications of the U. S. Geol. Survey, by Dr. C. W. Hayes, on the Georgia area; and to the valuable volume on "Manganese," by Dr. R. A. F. Penrose, issued by the Arkansas Geol. Survey, all of which have been freely used in the preparation of this paper.

† "Manganese: Its Uses, Ores and Deposits," *Annual Report of the Arkansas Geol. Survey*, 1890, vol. i., p. 417 *et seq.*

‡ "The Paleozoic Group," *Geol. Survey of Georgia*, 1893, pp. 190-209.

§ *Trans.*, xxx., 403-419 (1901).

talline Area; (c), the Paleozoic Group or Area. The areas are sharply marked one from the other by two strongly defined structural lines. First, the fall-line, which crosses the State in a slightly south of west direction, passing through or near the cities of Augusta, Macon and Columbus, separates the Coastal Plain sediments on the southeast from the Crystalline area. And second, the Cartersville overthrust fault separates the Crystalline area from the Paleozoic area on the northwest. The position of both the fall-line and the Cartersville fault are indicated on the sketch-map (Fig. 1) by the heavy broken lines.

The manganese-ores are limited in occurrence to the northern part of the State, distributed irregularly to some extent over parts of both the Paleozoic and Crystalline areas. The commercially important deposits are confined to the northwestern part of the State in the Paleozoic area. Small shipments, not amounting to more than a few tons in all, have been made of ores mined in different parts of the Crystalline area. No deposits of manganese are yet known to occur within the limits of the Coastal Plain.

Fig. 1, a sketch-map of the northern half of Georgia, shows the distribution of the manganese-deposits in the State, indicated by the solid black areas. The map further shows the outline of the areas covered by the Crystalline and Paleozoic rocks, and the extreme northwestern part of the Coastal Plain along the fall-line. That part of the Coastal Plain indicated on the map has its greatest width in a north-south direction along the eastern margin, extending south from Augusta.

The manganese-deposits of the two geologically different areas are best considered separately under I. The manganese-deposits of the Paleozoic area; and II. The manganese-deposits of the Crystalline area.

## I. THE MANGANESE-DEPOSITS OF THE PALEOZOIC AREA.

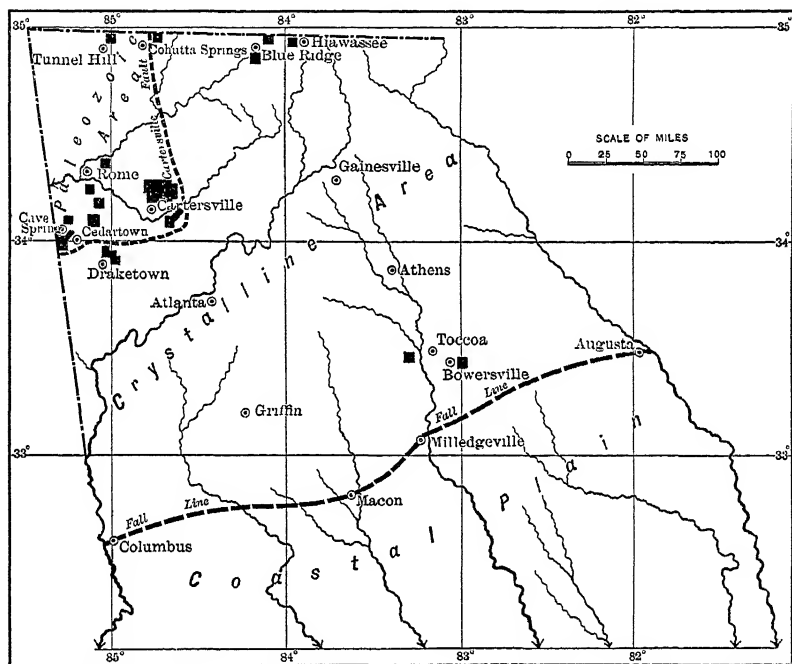
### *Résumé of the Geology of the Paleozoic Area.*

*Position.*—The position of the Paleozoic area is shown on the accompanying map (Fig. 1). The area includes the ten northwest counties of the State, and is separated on the east and south from the Crystalline area by the Cartersville over-

thrust fault. It forms a part of the southern extension of the great Appalachian valley southwestward into Alabama.

*Topography.*—The region as defined above is a long, narrow belt in which the valley-type predominates, and the axis of which has a general northeast-southwest trend. When viewed in detail, it is observed to be composed of numerous subordinate valleys, separated by more or less extensive parallel ridges, whose axial directions are coincident with the general trend of

FIG. 1.



Sketch-Map of a Part of Georgia, Showing the Distribution of the Manganese-Deposits (Represented by the Black Areas).

the valley-province. This ridge-valley type of topography bears a definite relation to the rock-structure of the area. The ridges mark the lines of more resistant rock, while the valleys are etched out of the soft shales and limestones. According to the character of the rocks composing the ridges, and the position of the beds with respect to the horizon, the ridges are high or low, rather broad and flat-topped, or narrow and sharp-crested. The Knox dolomite, one of the most persistent formations in the area, and of intermediate hardness, forms a

plateau of moderate elevation (between 900 and 950 ft. above mean-tide level) whose surface is gently undulating, usually not marked by any sharp ridges or peaks.

Traces of at least three rather distinct base-leveled plains appear in the region. According to Hayes\* the highest and earliest one of these plains was probably formed during Cretaceous time, and the period of rest during which the atmospheric forces were operative is believed to have been much longer than that of the formative period of either one of the subsequent plains. The present streams were revived by the recent uplift, and they are now engaged in sinking their channels in the surface of the last base-level.

*Stratigraphy.*—The rocks of the region range in age from lower Cambrian to Carboniferous, and they include slates, limestones, shales, sandstones and conglomerates. No igneous rocks are yet known to occur within the limits of the area. The manganese-deposits are limited to the residual decay resting on and derived from only three of the formations, namely, the Weisner quartzite, Beaver limestone and Knox dolomite. These formations are described in sufficient detail under the Cartersville and Cave Spring districts of this paper, and they need not be repeated here. (See Fig. 2.)

*Structure.*—The region is one in which the strata have been thrown into great northeast-southwest folds from horizontal pressure applied in a northwest-southeast direction. In addition to the folding, continuation of the same compressive forces resulted in fracturing and faulting the strata over most of the area. To the northwest of the Coosa valley the area is one of open folds, and faulting is less conspicuous. Folds of the anticlinal, synclinal and monoclinal types are represented in many examples of northeast-southwest-trending ridges preserved in the harder and more resistant rocks.

In the region to the south and southwest of Rome, the structure is more complicated, largely by reason of the folding having been quite or entirely obliterated by subsequent faulting, and by deposition-overlaps and abrupt lithologic changes.† Two classes of faults, which differ materially from each other,

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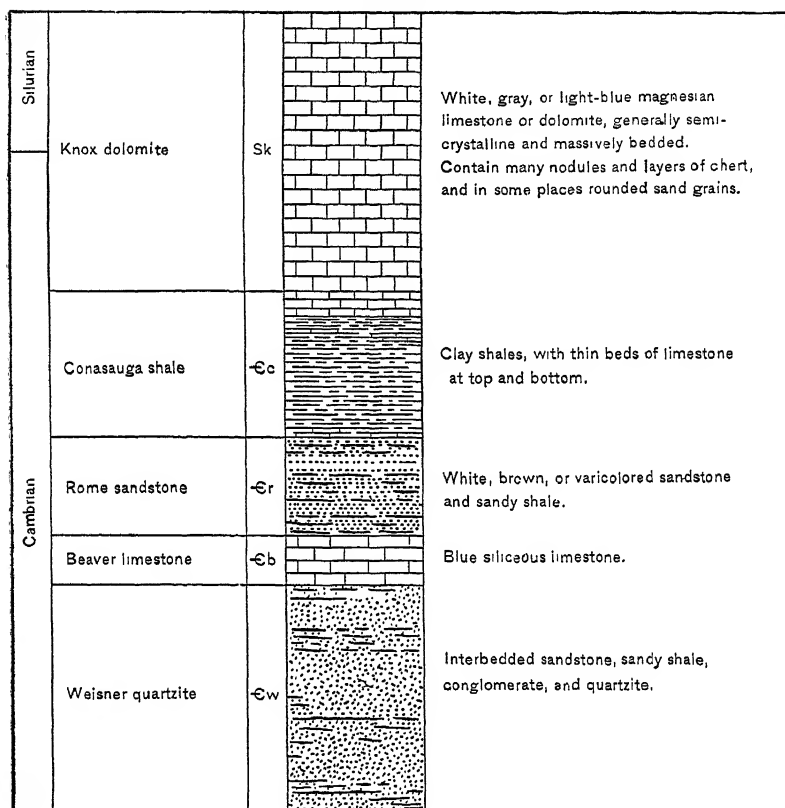
\* *Sixteenth Annual Report U. S. Geol. Survey*, 1895, Part III., pp. 553–554.

† Hayes, C. W., *Bulletin, Geol. Soc. of America*, 1894, vol. v., p. 472.

characterize the area. These are designated by Hayes\* as (a) major-thrust faults, and (b) minor-thrust faults.

The major-thrust faults are characterized by great horizontal displacement and low inclination of the fault-plane. Three

FIG. 2.



Generalized Stratigraphic Section of the Georgia Manganese-Area. (Adopted from Hayes.) Scale, approximately, 1 in. = 2000 ft.

faults of this type have been recognized and described by Hayes† in this area, namely, the Coosa, Rome and Cartersville overthrust faults. In the case of the Cartersville fault, rocks of probable Algonkian age and belonging to the Crystalline-

\* *Op. cit.*, pp. 557-560.

† Hayes, C. W., "The Overthrust Faults of the Southern Appalachians," *Bulletin, Geol. Soc. of America*, 1891, vol. ii., pp. 141-154; "Geology of a Portion of the Coosa Valley in Georgia and Alabama," *Bulletin, Geol. Soc. of America*, 1894, vol. v., pp. 465-480.

metamorphic area of the State, have been overthrust upon Silurian rocks of the Paleozoic area. (See Fig. 3.) These faults bear no relation to the manganese-deposits, and, therefore, need not be more fully described in the present connection.

The minor-thrust faults characterize the southern part of the area, especially of the area immediately south of Rome and in the vicinity of Cave Spring, and are of the ordinary Appalachian type. (See map of the Cave Spring district, Fig. 9.) They have an approximate north-south direction, intersecting the main axis of the region at angles of 30 to 40 degrees, or thereabouts. In length they vary from 3 to 8 miles, and they cut the strata at close intervals into narrow strips, forming monoclinals, which dip steeply toward the east. The faults of this type in the region already mentioned result in long and narrow strips of the underlying Conasauga shales, which form the narrow valleys penetrating southward into the Knox dolomite plateau.

For the reason that the two types of faults here distinguished are seldom found intersecting each other, the faulting is inferred to belong to different and, therefore, distinct periods of disturbance.

#### *The Cartersville District.*

One of the principal manganese-producing districts in the Southern Appalachians is in the vicinity of Cartersville, Bartow county, Georgia, about 50 miles northwest of Atlanta. Manganese-mining in this district dates back as early as the year 1866, when 550 tons of the ore are reported to have been mined.\* From that time to the present the Cartersville district has been one of the three principal producers of manganese in the United States. Excepting the Cave Spring district, the entire production of manganese in Georgia has been from this district.

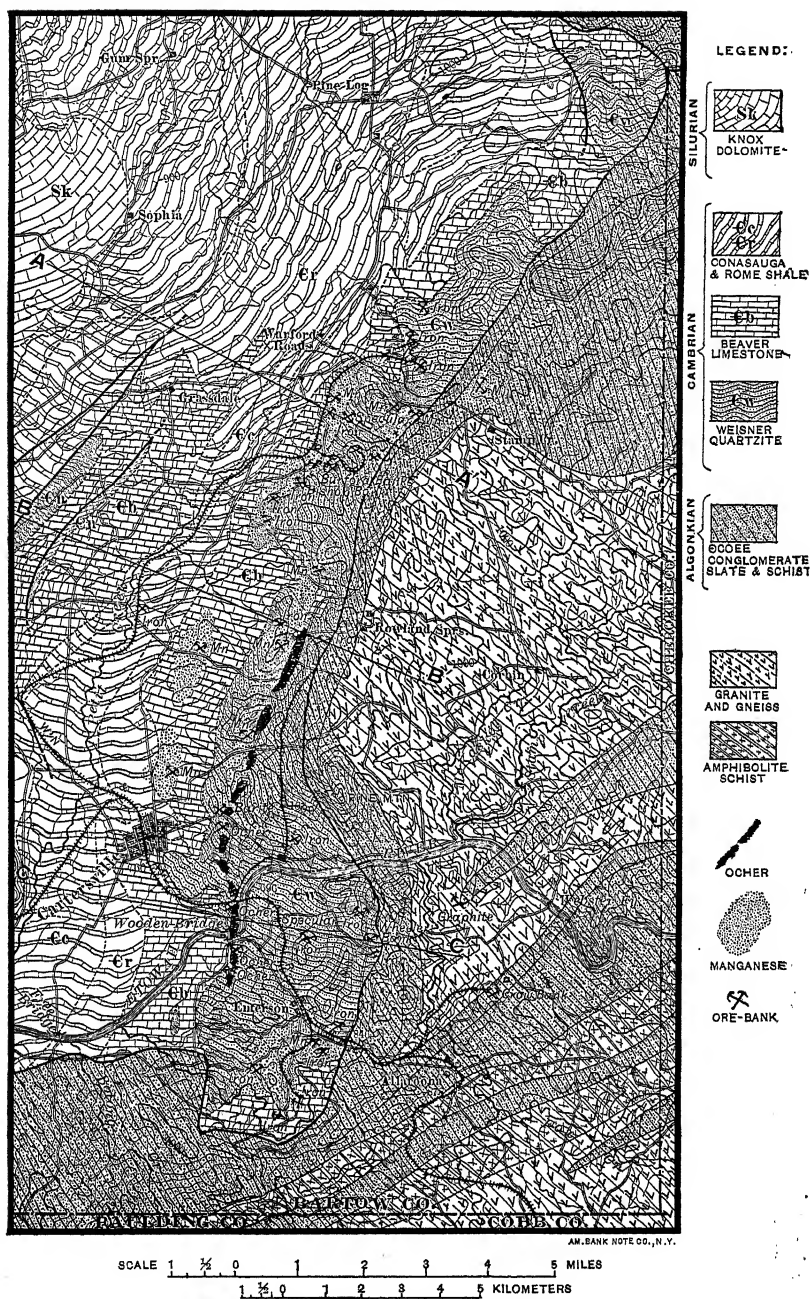
The older crystalline and metamorphic rocks occupy the east half of the area shown on the map (Fig. 3). The Paleozoic formations of the valley province occupy the west half of the mapped area. The line separating the two groups is an irregular one, marking the position of the Cartersville fault.

The succession of formations in ascending order, on the west side of the Cartersville fault, is as follows :

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\* *Mineral Resources of the United States*, 1885, p. 329.

FIG. 3.





Silurian, . .	Knox dolomite. Cherty magnesian limestone.
	Conasauga shale. Olive clay shale, chiefly.
	Rome shale and sandstone. Purple, white, green and brown sandstone, and interbedded sandy shale.
Cambrian, . .	Beaver limestone. Blue siliceous limestone.
	Weisner quartzite. Quartzite, coarse conglomerate and micaceous shale.

Except the Knox dolomite, all the formations shown on the map to the west of the Cartersville fault belong to the middle and lower Cambrian. The Knox dolomite is here included entirely in the Silurian, although strong reasons appear for grouping the lower portion of this formation with the Cambrian. In Figs. 2, and 9, the Knox dolomite is designated as partly Cambrian and partly Silurian.

*The Paleozoic Rocks of the Cartersville District on the West Side of the Fault.*

*The Weisner Quartzite.*—The Weisner quartzite is in contact on the east side with the Cartersville fault. The principal area of the quartzite occupies the middle portion of the map as an irregular strip, having an approximate north-south extension of about 15 miles and, in width, varying from 1 to 3 miles. It is composed principally of a fine-grained quartzite, with additional bands of a fine-grained conglomerate and micaceous shales. Wherever exposed, the formation shows evidence of intense folding, fracturing and crushing. The absence of satisfactory exposures, added to the complex folding of the beds, prevents an accurate estimate of its total thickness, but it is probably not less than 2000 to 3000 ft. thick in this locality, as stated by Hayes.

Hand-specimens of the Weisner quartzite, carefully collected by the writer from the numerous outcrops of the entire area exposed in Georgia, were thoroughly mixed and prepared as one sample, a representative part of which yielded, at the Pratt laboratory, Atlanta, the results shown on page 215.

*The Beaver Limestone.*—The main belt of the Beaver limestone lies along the western base of the Weisner quartzite ridges. A second, but smaller, area of the limestone extends from Grassdale southward to the line of the Western and Atlantic railroad, and is indicated near the western margin of

*Analysis of Weisner Quartzite, Cartersville District.*

SiO <sub>2</sub> , . . . . .	90.36
TiO <sub>2</sub> , . . . . .	0.07
Al <sub>2</sub> O <sub>3</sub> , . . . . .	1.52
Fe <sub>2</sub> O <sub>3</sub> , . . . . .	0.57
CaO, . . . . .	0.27
MgO, . . . . .	0.27
MnO, . . . . .	None
Na <sub>2</sub> O, . . . . .	0.43
K <sub>2</sub> O, . . . . .	0.16
H <sub>2</sub> O at 100° C., . . . . .	None
H <sub>2</sub> O above 100° C., . . . . .	0.31
FeS <sub>2</sub> , . . . . .	1.50
BaSO <sub>4</sub> , . . . . .	4.46
Total, . . . . .	99.92

the map. Exposures of the fresh limestone are rarely seen, since it is readily soluble, resulting in the insoluble residue forming a thick mantle of deep-red soil. The formation is readily traced, however, from the resulting red soil. Fragments of the quartzite in all stages of decay, derived from the adjacent higher quartzite ridges on the east, are admixed in some quantity with the decay derived from the limestone. The few exposures of the limestone met with indicate a semi-crystalline, gray, magnesian limestone, containing, as Hayes states, occasional masses of chert, becoming shaly in places. Hayes\* estimates the thickness of the limestone to be not less than 800 to 1200 ft.

These two formations, the Weisner quartzite and Beaver limestone, are of considerable importance as ore-producing formations in the Cartersville district, as a majority of the ore-deposits are associated with them. The manganese-ores occur with about equal frequency in the residual decay of the two formations.

*The Conasauga Shale.*—Next above the Beaver limestone is the great thickness of the Rome and Conasauga sandstone and shale (Oostanaula shales of Spencer). The main outcrop of the shale occupies the northwestern part of the mapped area. A continuous band of the exposed shale extends southward from the main outcrop to the line marking the position of the Cartersville fault. The southern half of the band,

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\* *Trans.*, xxx., 406 (1901).

which extends several miles in a north, south and west direction from Cartersville, is greatly widened.

Petit creek, one of the principal streams in the area mapped, takes its rise in the Conasauga shales some 10 miles north of Cartersville, and maintains its entire course southward to the Etowah river on the soft shales.

In lithologic character, the shales vary from very fine-grained aluminous or clayey rocks to somewhat siliceous shales and sandstone, with the aluminous type predominant. Numerous exposures show interlayered thin- and thick-bedded limestone with the shale. In color, the shales vary from light-drab and yellow to dark-blue slaty, best described, as a whole, as olive shales. The weathered shale is usually tinted some light shade of red, in marked contrast to the deep-red decay of the limestone. The shales are much fractured and crushed from the effects of intense pressure metamorphism. The thickness of the formation is placed between 1500 and 3000 ft.

The following analyses give a general idea of the chemical composition of the shales in this region:

*Analyses of Middle Cambrian Shales, Cartersville District.*

	I.	II.	III.	IV.
SiO <sub>2</sub> (free sand), } .	55.02	52.82	{ 62.30	39.20
SiO <sub>2</sub> (combined), }			{ 9.30	19.40
Al <sub>2</sub> O <sub>3</sub> , . . . . .	21.02	26.17	11.50	18.05
Fe <sub>2</sub> O <sub>3</sub> , . . . . .	5.00	9.46	5.59	8.81
FeO, . . . . .	1.54	.....	.....	.....
MnO, . . . . .	trace	.....	0.60	.....
CaO, . . . . .	1.60	trace	none	none
MgO, . . . . .	2.32	1.08	1.30	1.55
Na <sub>2</sub> O, . . . . .	0.81	0.20	0.35	0.33
K <sub>2</sub> O, . . . . .	3.19	2.71	4.20	4.63
H <sub>2</sub> O at 110° C., . . .	2.44	{ 0.23	0.15	0.40
		{ (hygr.)	(hygr.)	(hygr.)
H <sub>2</sub> O above 110° C., .	5.65	7.00	3.80	7.60
TiO <sub>2</sub> , . . . . .	0.65	.....	1.10	0.68
P <sub>2</sub> O <sub>5</sub> , . . . . .	0.06	.....	.....	.....
BaO, . . . . .	0.04	.....	.....	.....
SrO, . . . . .	trace	.....	.....	.....
Li <sub>2</sub> O, . . . . .	0.03	.....	.....	.....
SO <sub>3</sub> , . . . . .	0.02	.....	.....	.....
Cl, . . . . .	trace	.....	.....	.....
CO <sub>2</sub> , . . . . .	0.83	.....	.....	.....
Carbonac. matter, . .	0.32	.....	.....	.....
Total, . . . . .	100.54	99.67	100.19	100.15.

I. Middle Cambrian shale. Coosa Valley, near Blaine, Cherokee county, Alabama. H. N. Stokes, analyst. *Bulletin No. 168, U. S. Geol. Survey, 1900*, p. 283.

II. Oostanaula (Conasauga) shales, about 2 miles northwest of Cartersville, Bartow county, Georgia. J. M. McCandless, analyst. "Paleozoic Group," *Geol. Survey of Georgia, 1893*, p. 285.

III. Light-colored hydro-mica shale on the ridge above the Etowah river iron bridge, south of Cartersville. On the border of the metamorphic zone. "Paleozoic Group," *Geol. Survey of Georgia, 1893*, p. 284. J. M. McCandless, analyst.

IV. Light-red shale, in the valley 1 mile southwest of Cartersville. "Paleozoic Group," *Geol. Survey of Georgia, 1893*, p. 284. J. M. McCandless, analyst.

*The Knox Dolomite.*—The Knox dolomite lies next above the Conasauga shales. The lower beds are probably Cambrian, but, owing to the paucity of fossils in them and the striking uniformity in lithologic character, the entire formation is here classed as Silurian. Hayes says:\*

"From the few fossils which have been found, it appears probable that a transition from Cambrian to Silurian occurs in the lower third of the formation, but it is generally impossible to determine this line of division. . . ."

It is a massively-bedded, semi-crystalline, gray magnesian limestone, containing abundant nodules and layers of chert, and is one of the most persistent formations in the Southern Appalachians.

Weathering of the dolomite, by removal in solution of the soluble calcium and magnesium carbonates, has covered the limestone surface with a prevailingly thick mantle of insoluble siliceous clays abundantly admixed with chert-fragments and masses. The residual clays derived from the dolomite are often light in color, containing comparatively small amounts of iron oxide. In many places, however, the clays derived from the limestone are deep-red ferruginous clays, containing much iron oxide. The proportion of clay to chert varies considerably, but the quantity of chert admixed with the clay is always large. It is by means of its residual material, especially the chert nodules and masses, that the Knox dolomite is often traced, for exposures of the fresh rock are seldom seen except along the stream-courses.

The magnesian limestone has an estimated thickness of 3000

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\* "Geologic Atlas of the United States," Rome Folio, *U. S. Geol. Survey, 1902*, p. 3.

to 5000 ft. As a producer of manganese-ores, the Knox formation is, perhaps, of less importance in the immediate Cartersville district than either the Weisner quartzite or Beaver limestone. Excepting the Cartersville district, the Knox dolomite is one of the most important ore-producing formations in the Georgia Paleozoic area, as the extensive accumulations of bauxite and a part of the iron- and manganese-deposits are associated with it.

The numerous chemical analyses made from specimens collected from the Knox formation over many localities in Georgia show its composition to vary within the following limits:

SiO<sub>2</sub>, 3.75 to 7.25; Al<sub>2</sub>O<sub>3</sub> and Fe<sub>2</sub>O<sub>3</sub>, 1.24 to 1.76; CaCO<sub>3</sub>, 34.07 to 53.44; and MgCO<sub>3</sub>, 36.32 to 55.74 per cent.

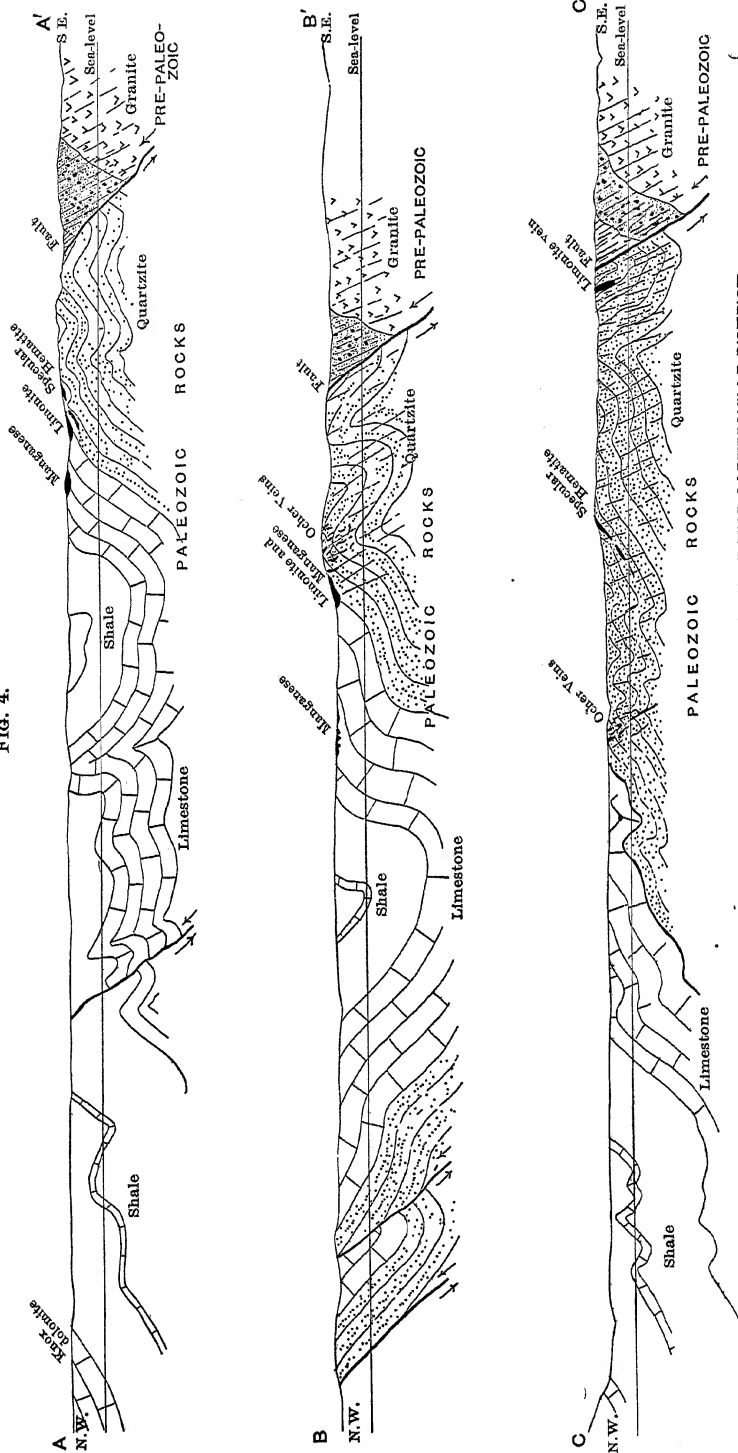
*The Older Crystalline and Metamorphic Rocks of the Cartersville District on the East Side of the Cartersville Fault.*

Several types of crystalline, metamorphic rocks are represented which show wide variation in composition and, probably, in age. Of these, the Corbin granite-area, which occupies the middle eastern portion of the map (Fig. 3), is the most extensive. As mapped, this granite mass is a roughly oval-shaped area extending from Stamp Creek P. O. on the north to the line of the Western and Atlantic railroad on the south, and continuing eastward into Cherokee county. The granite is a coarse-grained porphyritic rock, presenting a distinct augen-gneiss facies in the border-portions. It is composed of large, microcline phenocrysts imbedded in a ground-mass of blue quartz, plagioclase feldspar, augite and mica. Its composition is shown in the following analysis, made by Dr. H. N. Stokes, of specimens collected by Mr. A. H. Brooks of the U. S. Geological Survey, 1 mile east of Rowland, Bartow county, Georgia:

*Analysis of Granite, Cartersville District.*

SiO <sub>2</sub> , . . . .	67.98	TiO <sub>2</sub> , . . . .	0.84
Al <sub>2</sub> O <sub>3</sub> , . . . .	14.84	P <sub>2</sub> O <sub>5</sub> , . . . .	0.34
Fe <sub>2</sub> O <sub>3</sub> , . . . .	1.00	MnO, . . . .	trace
FeO, . . . .	3.15	BaO, . . . .	0.20
MgO, . . . .	0.91	SrO, . . . .	trace
CaO, . . . .	2.17	Li <sub>2</sub> O, . . . .	trace
Na <sub>2</sub> O, . . . .	2.66	S, . . . .	0.08
K <sub>2</sub> O, . . . .	4.76	Cl, . . . .	trace
H <sub>2</sub> O+, . . . .	0.14	F, . . . .	trace
H <sub>2</sub> O-, . . . .	0.49	C (Graphite), . . . .	0.21

FIG. 4.



SECTIONS ON LINES INDICATED IN FIG. 3, SHOWING GEOLOGIC STRUCTURE OF THE CARTERSVILLE DISTRICT.

Scale, horizontal and vertical 1 in. 5000 ft.

(From the paper of C. W. Hayes, *Trans.*, xxx., 407.)

Brooks has given the following petrographic data on the granite from this locality:

"Contains microcline, some plagioclase, abundant pyroxene, partly altered into chiefly uraltite and chlorite, some biotite with frequent inclusions of rutile, much blue vitreous quartz, apatite, zircon and magnetite."\*

In places, the border-portion of the granite mass is overlapped by a coarse feldspathic conglomerate whose mineral constituents were evidently derived from the granite, since the microcline and the blue quartz of the granite enter largely into the composition of the conglomerate. In other places a series of black graphitic slates are in contact with the granite. No fossils are known to occur in the conglomerates and slates, and on account of their appearance of extreme age, Hayes† has grouped them as Algonkian (Ocoee).

To the south of the Corbin granite-area, the conglomerates and slates increase in metamorphism and apparently pass into schists and gneisses, whose origin, whether igneous or sedimentary, is unknown.

The extreme southeast corner of the map (Fig. 3) comprises narrow belts of granite, gneiss and hornblende (amphibolite) schist. According to Hayes,‡ both diabases and diorites are represented in the area. Diorite is regarded as the commonest type of rock, and it is now mostly altered into the amphibolite schist.

### *Structural Features.*

It is only necessary in this paper briefly to call attention to the broader structural features of the district, since the manganese-deposits are limited to the residual clays of several of the formations, and only in a very general way do they show any relations to the structure. In his paper on the iron ore-deposits of the district, Hayes has discussed in some detail and in a most excellent manner the structure of the region, to which paper the reader is referred for further details.§

The area has been one of intense and prolonged disturbance (compressive forces) operating in a northwest-southeast direction, resulting in profound alteration of the rocks,—their fold-

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\* Clarke, F. W., *Bulletin, U. S. Geol. Survey*, No. 168, 1900, p. 55.

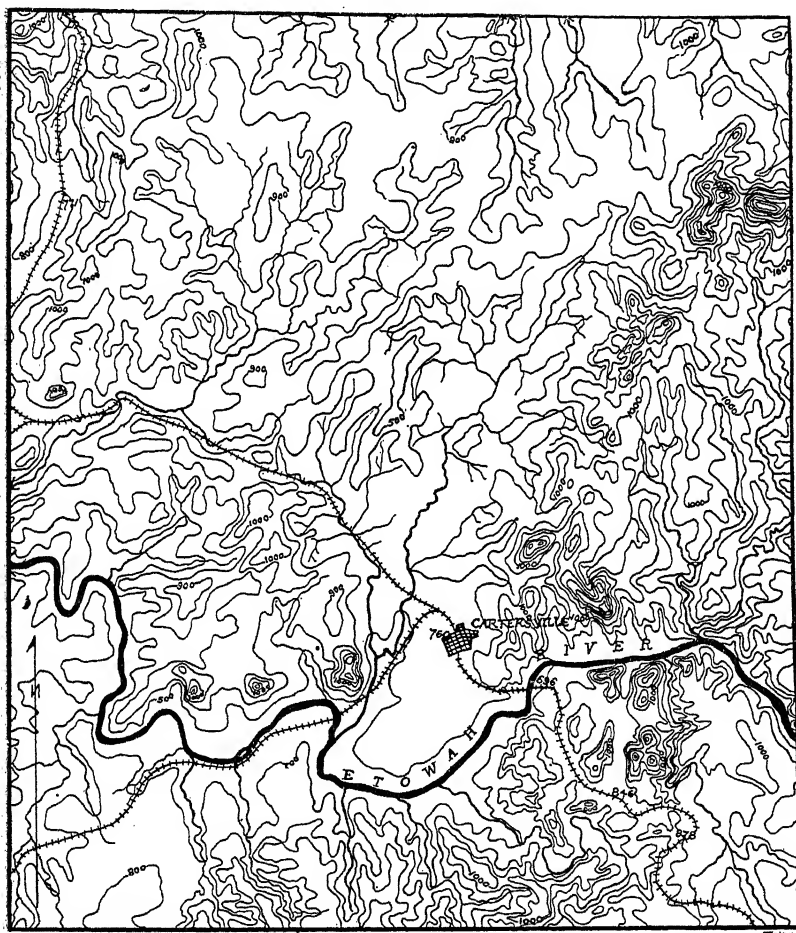
† Hayes, C. W., *Trans.*, xxx., 408 (1901).

‡ *Ibid.*, p. 408.

§ *Op. cit.*, pp. 403-419.

ing, crushing and fracturing. In places, some of the rocks, originally unlike, are so profoundly altered that they are with great difficulty distinguishable at present. On the east side of the fault-line the rocks are mashed and squeezed, and the

FIG. 5.



0 1 2 3 4 5  
Scale of Miles.

Topographical Map of the Cartersville District, Georgia. (Cartersville Topographic Sheet, *U. S. Geol. Survey.*) Contour-interval, 100 ft.

slaty and schistose structures strongly developed, while on the west side the rocks are complexly folded and fractured. In addition to the folding, the Weisner quartzite especially has been much crushed and in places brecciated, probably indicat-



ing faulting. Some of the more massive formations, such as the Knox dolomite, resisted the folding to a greater degree than others.

The overthrust fault which separates the older crystalline rocks from the Paleozoic sediments is the principal structural feature of the region. Its position is shown on the accompanying map (Fig. 3), where it is observed to pass some distance to the east and south of Cartersville, from which town it derives its name. Its line of contact is here marked by the rocks of the Ocoee series on the east side, brought next to the rocks of the Cambrian on the west side.

*The Manganese Ore-Deposits of the Cartersville District.*

*Mode of Occurrence of the Ores.*—The manganese-ores occur imbedded in the heavy mantle of residual material derived from the decay of the Beaver limestone and the Weisner quartzite, and they have nearly equal distribution in the decay derived from the two formations. The decay of the Weisner quartzite is a gray to yellow siliceous clay admixed with fragments of the quartzite in all stages of decay; that of the Beaver limestone is a deep-red clay, less siliceous than that derived from the Weisner formation, admixed with some chert-fragments; and, near its eastern margin in contact with the Weisner quartzite, additional fragments of the latter rock are found.

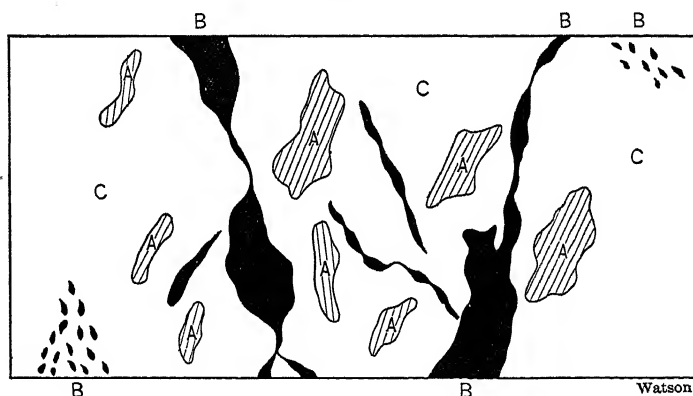
The ore is distributed through the clays in an extremely irregular manner in the form of pockets or lenticular masses, rarely as distinct beds; veins and stringers cutting the clays in all directions; as single nodules or concretionary masses assembled in the clays; and as small disseminated grains scattered through the clay. In places, the ore-distribution in the clays conforms in a general way to the bedding of the inclosing clays; usually, however, this is obscured and the ore-bodies indiscriminately cut in all directions. The pockets vary widely in size and number. They range in size from mere nests to bodies 6 and more ft. thick and more than 30 ft. long, and in extreme cases may yield several hundred tons of ore. Rarely are they composed of solid ore free from the surrounding clay, the usual form being that of somewhat thickly-studded nodules in the clays. They may occur close together, or far apart, and are usually not in any way connected; although in many cases

irregular stringers and small veins are observed to lead from one pocket to another. (See Figs. 6 and 7.)

The ores are never entirely free from inclusions and admixtures of the inclosing clays, a condition which naturally results from their method of accumulation. Some of the ores are, of course, freer from these mechanical impurities than others, and in the purest ore the included grains of silica and the adhering clay are reduced to a minimum. The proportion of clay to ore is usually larger than in the closely associated brown iron-ores.

*Depth of the Residual Decay.*—The depth of the rock-decay varies greatly, and it is dependent, other things being equal,

FIG. 6.



Section in One of the Openings on the Blue Ridge Mining Co.'s Property, near Cartersville, Georgia, Showing the Mode of Occurrence of the Manganese-Ore in the Residual Clay.

A, fragments and masses of partially decayed rock ; B, manganese-ore ; C, residual clay.

Horizontal and vertical scale, 1 in. = 17 ft.

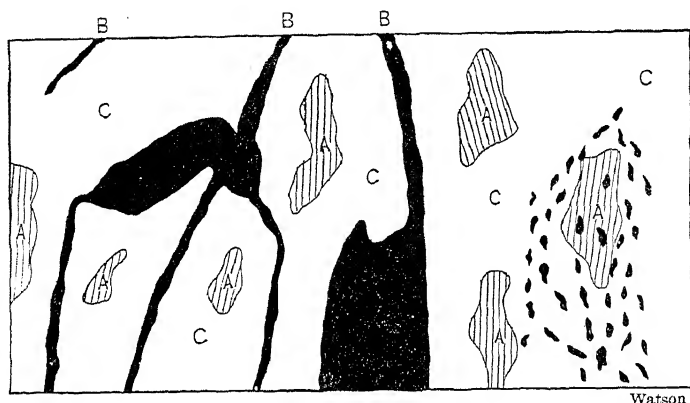
on the character and composition of the rock, and on the disposition of the rock-strata. The quartzite of the Cartersville district has been much broken and crushed, and it is thrown into a series of narrow, more or less steep folds or ridges, whose surface is everywhere covered to some depth with its residual decay. Reefs of the hard and fresh rock are often exposed along the crests of the ridges, and to some extent exposures are frequent near the tops and along the steeper slopes of the ridges.

Shafts have been sunk in several places to a depth of several hundred feet without piercing the bed-rock. Depths of 100 ft.

and more in the residual mantle are common in the district. In the Chumler Hill section, about 8 miles northeast of Cartersville, several shafts have been sunk to a depth of more than 80 ft. in manganese-mining without encountering the bed-rock. (See Fig. 11.)

*Kinds of Ore.*—Only the oxides of manganese occur in the Cartersville district. Of these, pyrolusite and psilomelane greatly predominate, with some manganite and braunite and much of the earthy oxide, wad. These different oxides cannot always be separated, but they usually occur admixed in vary-

FIG. 7.



Section in One of the Openings at the Dobbins Mine, near Cartersville, Georgia, Showing the Occurrence of Manganese-Ore in the Residual Clays. (Modified from Penrose.)

A, fragments and masses of partially decayed rock ; B, manganese-ore ; C, residual clay.

Horizontal and vertical scale, 1 in. = 10 ft.

ing proportions. With the exception of the mineral wad, the ore is usually partially or entirely crystalline, of a dark steel-blue color, and the nodular type which prevails nearly always displays the complete or partially layered or concentric structure of concretionary masses.

*Associated Ore-Deposits.*—Extensive deposits of brown iron-ore and gray or specular hematite, yellow ocher, and to a less extent barite and bauxite occur somewhat closely associated with the manganese-ores. Of these the deposits of iron-ore and yellow ocher have been extensively mined. The bauxite and barite are of less importance, since the former is only

sparingly found within the limits of the district, and the latter, while more abundant, is not sufficiently concentrated to admit of profitable working. These are again referred to at some length in a subsequent part of this paper discussing the origin of the manganese-ores.

*Manganiferous Iron-Ores.*—The beds of brown iron-ore, which is the prevailing type of iron-deposit in the district, are usually distinct from, though occurring in close relation with, the manganese-ores. The oxides of iron and manganese are often found admixed in different proportions in the same bed. Between the two extremes of pure iron-ore and pure manganese-ore occur all gradations in the admixture of the two oxides. In some of the beds the two materials are homogeneously mixed, giving the appearance of a manganese-ore when the iron is present in small quantity, and of the usual brown hematite-ore when manganese is in small quantity. In still other cases the beds of iron-ore are found incrustated in places at the surface for only a slight depth, with the oxide of manganese, which on being opened prove to be good deposits of iron-ore and not manganese.

Analyses of the iron-ores invariably show small percentages of manganese, and conversely the manganese-ores show varying percentages of iron, with intermediate gradations in which the two oxides are present in nearly equal amount, forming a good grade of manganiferous iron-ore. These gradations are shown to some extent in the following partial analyses of samples of the ores from the district:

*Analyses of Cartersville Iron- and Manganese-Ores.*

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Manganese, . . .	41.98	36.00	25.09	15.26	60.61	2.30	54.94	56.40
Iron, . . . .	16.22	16.88	29.17	39.25	1.45	52.02	3.62	1.29
Phosphorus, . .	0.227	0.14	0.155	0.193	0.052	0.24	0.034	0.158

*Chemical Composition.*—The hundreds of commercial analyses of the Cartersville manganese-ores indicate that the ores do not differ essentially from similar high-grade ores occurring elsewhere. The average in metallic manganese in the better-grade ores is uniformly high, with correspondingly low iron, silica and phosphorus. Silica will usually average low in those cases where the ore has been properly cleansed. It rarely

ranges above 10 per cent., and is usually much below this, averaging from 2 to 5 per cent. Of course, in many of the lower-grade ores the silica will average considerably above 10 per cent. The phosphorus in the Cartersville ores is rarely high enough to detract from the value of the ore. The average in this ingredient for the better-grade ores of the district is from 0.10 per cent. to 0.15 per cent., rarely rising above 0.25 per cent.

*Individual Properties.*—No attempt will be made in this paper to discuss or describe individual properties. These will be described in detail in a forthcoming report by the writer, to be published by the Geological Survey of Georgia. Among some of the more important properties in the district which have yielded much ore of superior quality may be mentioned the Blue Ridge Mining Company (formerly known as the Etowah Mining Company), the Dobbins, the Georgia Manganese and Iron Company, the Milner-Harris, the Satterfield, the John P. Stegall, the Chumler Hill and the Southern Mining Company. Innumerable single lots of land, distributed over all parts of the district and owned by different individuals, have produced large quantities of high-grade ore, but the lack of space forbids their mention by name.

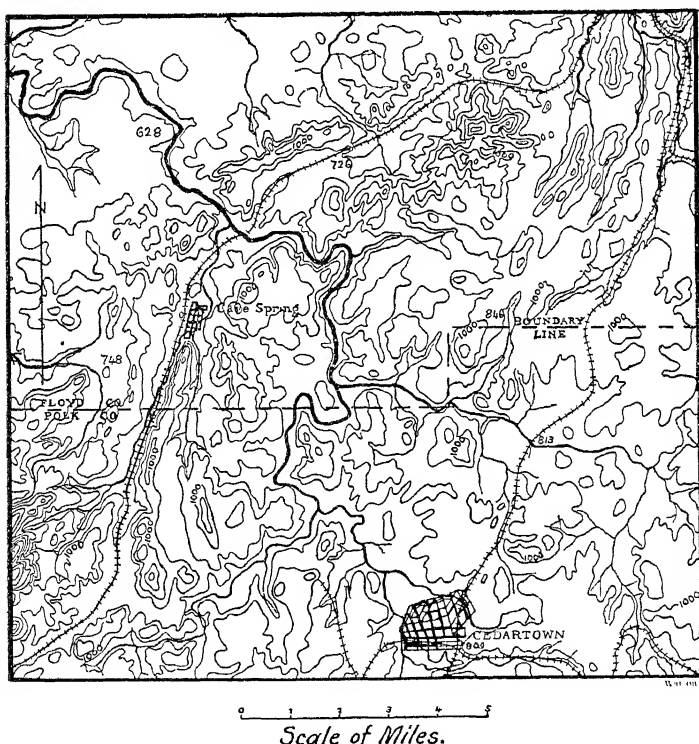
### *The Cave Spring District.*

The Cave Spring district occupies the southwest part of Floyd county and the northwest corner of Polk county, Georgia. The town of Cave Spring, from which the district takes its name, is within 6 miles of the Alabama-Georgia line, about 15 miles southwest of Rome. The ore-deposits of manganese extend about 5 miles south of the town into Polk county, and continue northeastward from the town for a distance of 7 or 8 miles in Floyd county. Practically the entire production of manganese-ores in Georgia has been from the Cartersville and Cave Spring districts.

*Topography.*—The topography of the Cave Spring area is shown on the accompanying map (Fig. 8). The region is one in which the topography is strikingly shown to be dependent on the structure and character of the underlying rocks. The strata are broken by numerous approximate north-south faults of the ordinary Appalachian type, resulting in monoclinical

blocks, which dip somewhat steeply to the east and southeast. Faulting has brought the underlying shale to the surface in contact with the overlying Knox dolomite, and valleys are etched out of the soft shale, penetrating to some extent the harder and more resistant Knox dolomite. These features are well developed in the vicinity of Cave Spring and are shown again in the northeast corner of the map (Fig. 9).

FIG. 8.

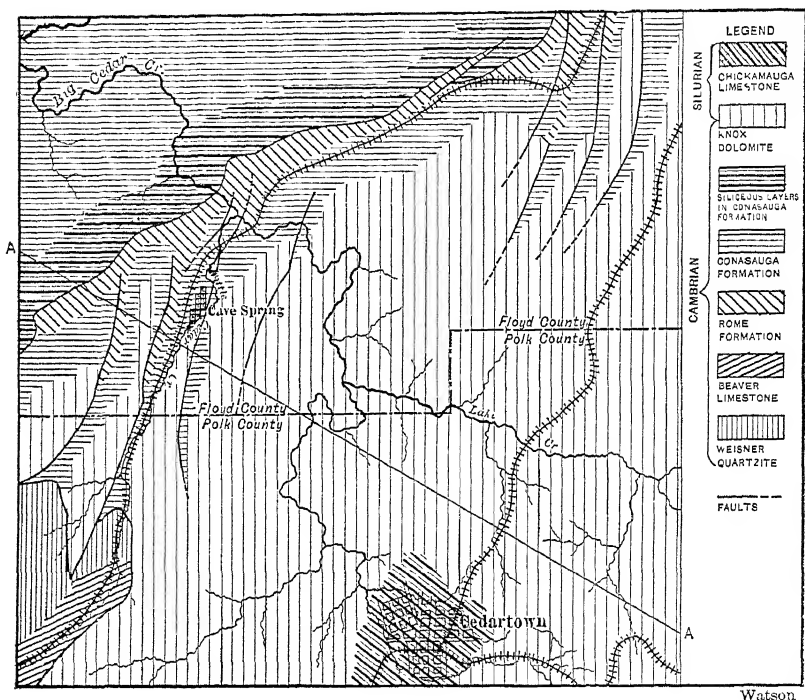


Topographical Map of the Cave Spring District, Georgia. (Topographic Sheet, Rome Folio, *U. S. Geol. Survey.*) Contour-interval, 100 ft.

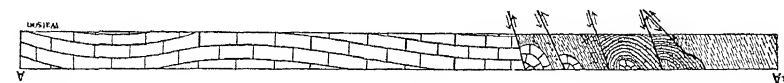
To the east of a line drawn through the town of Cave Spring the area is underlain by the Knox dolomite. Surface-erosion is relatively slower over this part of the area, on account of the large proportion of chert contained in the dolomite, than in other parts underlain by softer and less resistant rocks. The surface is generally hilly and stands several hundred feet above the valley-floors of softer rock. It forms a broad plateau, aver-

aging slightly below 1000 ft. in elevation. In the extreme south-west corner of the map (Fig. 9), the hard and resistant massive Weisner quartzite is exposed as a high ridge, known as Indian mountain, which lies almost entirely in Alabama. Traces of the same base-levels are preserved in the rocks of this area as those mentioned in the region farther north.

FIG. 9.



Watson



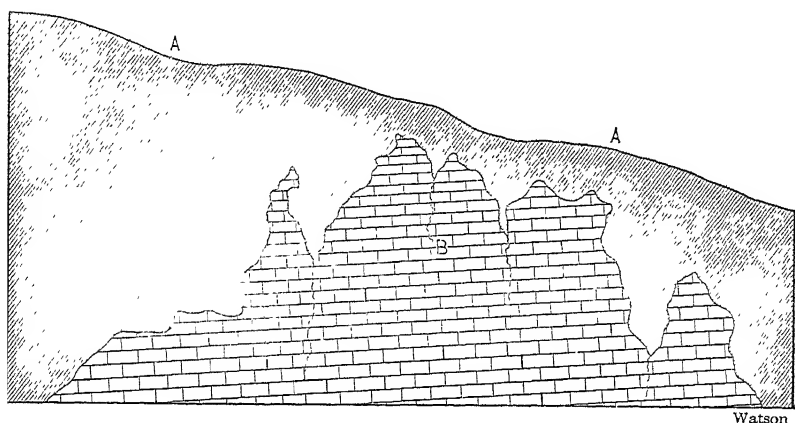
Geological Map and Section of the Cave Spring District, Georgia. (Rome Folio, *U. S. Geol. Survey.*)

*Stratigraphy.*—The same rock-sequence is observed here as in the Cartersville area. (See Fig. 2.) The same formations are represented in the two areas, with only slight variations indicated in the general character and thickness of the rocks. These formations have been described in sufficient detail under the Cartersville district, and will not be repeated here. The

Knox dolomite of the Cave Spring district needs further description, since it is the only formation in the area with which the manganese-ores are associated.

The Knox dolomite is vastly the most extensive formation in the district. The percentage of chert is much larger, and the cherty masses and fragments are larger in size than elsewhere for this formation. In places, the limestone appears to be largely replaced by layers of the massive chert. Its surface is very generally strewn with the chert, and the deep-red clays derived from the limestone are heavily charged with the chert-fragments in all stages of decay.

FIG. 10.



Section in Knox Dolomite, 2 miles East of Kingston, Georgia, Illustrating Weathering of the Magnesian Limestone. (Modified from Spencer.)

A, residual clay; B, fresh magnesian limestone.

*Structure.*—Referring again to the map (Fig. 9), it is observed that nearly the entire southeast part is occupied by the massive Knox dolomite, which resisted the sharp folding manifested in some of the other formations. The remaining part of the map indicates numerous faults cutting the strata. These are mostly of the minor-thrust type, intersecting the Knox dolomite and the Conasauga shale, and they expose at the surface the soft shales in long and narrow north-south strips, forming narrow valleys among the dolomite hills. The fault-blocks overlap each other, with rather steep dips toward the east and southeast.

*Character of the Residual Decay.*—The manganese-ores of the



Cave Spring area are entirely limited to the residual clays derived from the decay of the Knox dolomite. The decay from this formation only will be considered. The ore-bearing clay is usually of a deep-red, chocolate or brown color, with lighter tints occurring. It is soft and plastic when wet, and is generally associated with much siliceous material in the form of chert, subject to wide variation from place to place. Its depth is variable and, in places, the moderately fresh cherty limestone is exposed as broken reefs on the ridge-tops. In other places on the ridge-tops, excavations in mining expose the hard rock at depths varying from 10 to 30 ft.; and, in still other places on the same ridges, rock is not encountered at depths of 50 to 60 ft.

Massive cherty layers or beds are of more frequent occurrence in the limestone of this area than in the same formation at other points in northwest Georgia. Here the chert attains considerable thickness, and, at times, almost entirely replaces the limestone. Usually the chert is a white or gray rock, sometimes brown, rarely black, and is often incrustated with a film or coating of the black oxide of manganese. The relation of the chert and limestone is well illustrated along the ridges to the southeast of the town of Cave Spring, where the rocks dip to the southeast, with the limestone forming the lower part of the hills and the chert the upper part.

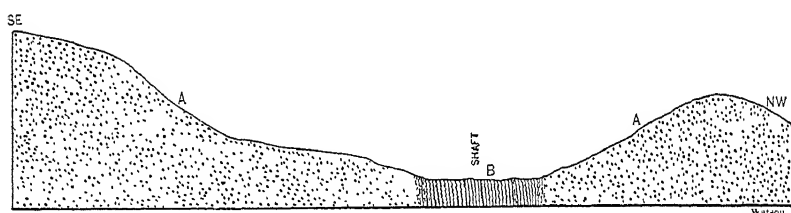
*Manganiferous Chert-Breccia.*—The very intimate association of the chert and ore has resulted in the formation of much breccia, composed of the angular chert-fragments cemented in a manganese-oxide matrix. This seems to have originated in most cases from the infiltration of surface-waters containing manganese in solution. The occurrence of the chert-breccia beneath the clay covering is excellently illustrated in some of the excavations on the ridge above the Cedar creek valley, and again in the openings on Reynolds mountain, 8 miles north-east of Cave Spring. (See Fig. 15.)

*Manganiferous Stained Chert.*—Where the chert is mixed in large proportion with the ore-bearing clays, it is often more or less stained with the manganese. This may occur as thin films or layers of the manganese oxide coating the loose chert-fragments, or as stringers or veinlets filling the cracks in the chert. The proportion of staining to chert is quite variable.

In many cases it amounts to only a mere film; in others, the veins filling the cracks are quite thick. This has resulted from the free percolation of manganese-bearing waters through the loose cherty clays. Sometimes, that part of the chert not impregnated has been entirely or partially removed by decay, leaving a mass of siliceous manganese-ore of various shapes.

*Occurrence of the Ores.*—The occurrence of the manganese in this area is closely similar to that in the Cartersville district, already described. (See Figs. 6, 7 and 15.) The ores differ principally in occurrence from those in the Cartersville district in (*a*) their intimate association with the cherty beds and clays of the upper part of the Knox dolomite, in (*b*) being limited to the residual clays derived from only one formation, and (*c*)

FIG. 11.



Section through the Chumler Hill Mine, Georgia, Showing the Mode of Occurrence of the Manganese-Bearing Clay. (After Penrose.)

A, sandstone; B, manganese-bearing clay.

Horizontal scale, 1 in. =  $\frac{1}{4}$  mile. Vertical scale, 1 in. = 400 ft.

being stratigraphically above those of the Cartersville district, occurring in the clays of the Knox dolomite and not found at all in the clays of the Weisner quartzite and Beaver limestone, which are the ore-bearing formations in the Cartersville district. The ore-occurrence in the residual clays is identical for the two districts. In the Cave Spring district the ore has been observed penetrating and filling the cracks in the partially decayed rock below in all directions with much the appearance of brecciated masses.

*Associated Ore-Deposits.*—Beds of brown iron-ore are frequently found in close relation with the manganese-ores. They are much more extensive and more constant than those of manganese, and the iron displays a greater tendency to a bedded form and a less tendency to the nodular form. The region in

the vicinity of Cave Spring and Cedartown is probably the largest brown iron-ore-producing area in the State. A few scattered deposits of bauxite occur, but this mineral is less closely associated with the manganese than are the beds of iron-ore.

*Chemical Composition.*—Only the oxides of the metal occur, the principal ones of which are psilomelane and pyrolusite. Admixed with these two oxides occur varying smaller percentages of several of the other oxides, especially braunite. The following chemical analysis of specimens of the purer ore collected from the mine of Major Couper, south of Cave Spring, and analyzed by Mr. Britton, will indicate the general character of the ores.

*Analysis of Manganese-Ore, Cave Spring.*

Metallic manganese, . . . . .	53.44
Ferric oxide, . . . . .	2.83
Barium oxide, . . . . .	8.62
Silica, . . . . .	7.79
Alumina, . . . . .	1.52
Lime, . . . . .	0.08
Phosphoric acid, . . . . .	0.147
Water, . . . . .	1.56
Oxygen with manganese, undet., etc., . . . . .	24.013
Total, . . . . .	100.000

*Other Manganese-Deposits of the Paleozoic Group.*

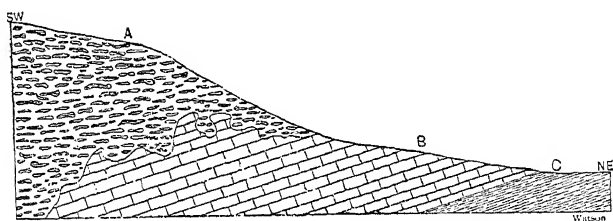
Under this heading are included certain centers about which are grouped a few scattered deposits of manganese-ores. These centers are found over parts of the northeastern, eastern and southern portions of the Paleozoic area. Many of the deposits have been worked to some extent, but in most cases the work has not progressed beyond the stage of test-openings. In some cases, no openings of any nature have been made, but strong surface-indications appear, which may or may not imply workable deposits below the surface. Small shipments of the ore have been made from a number of the openings, but as yet these scattered accumulations of the ore have proved of little or no commercial importance. Further developments in some of the localities may, perhaps, lead to important concentrations of workable ore. The mode of occurrence, association and the mineral character and form of the ores are the same as described in the Cartersville and Cave Spring districts.

The following localities include the list of these scattered ores: In the vicinity of Ligon P. O., in the extreme southwest corner of Bartow county, about 12 miles west of Cartersville; near Rome and Lindale, in Floyd county; in Big Texas Valley, 12 miles northwest of Rome, in Floyd county; the Barnsley estate and vicinity, in the northwest part of Bartow county and the adjacent part of Floyd county, 17 miles northwest of Cartersville; and the Tunnel Hill district, in Whitfield and Catoosa counties.

Attention is briefly called to the deposits in two of these localities.

*The Lindale Deposit.*—Extending southward from Lindale is a long, narrow strip or belt of shale which marks the position

FIG. 12.



Section through the Barnsley Tract, Georgia, Showing Manganese-Bearing Chert-Bed. (Modified from Penrose.)

A, chert and cherty limestone; B, limestone; C, shale.

Horizontal scale, 1 in. = 500 ft. Vertical scale, 1 in. = 200 ft.

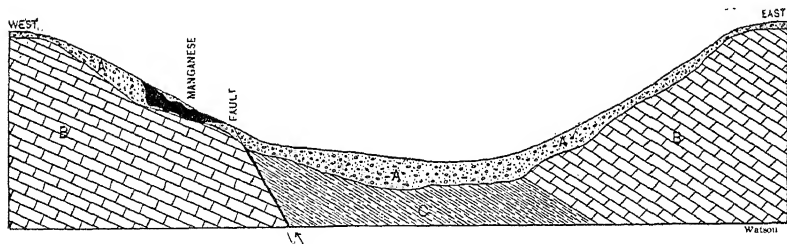
of a valley for several miles in a north-south direction. The western margin of the valley marks the position of a minor-thrust fault, and to the eastward across the valley, about  $\frac{1}{4}$  of a mile, the shales are overlain by the heavy beds of cherty limestone. The ridge is a low one, averaging more than 100 ft. in elevation along its highest portions, and its surface is irregular and broken by erosion. In places, it is covered with chert-fragments of all sizes in various stages of decay. (See Figs. 13 and 14.)

About  $\frac{3}{4}$  of a mile south of Lindale, on the west slope of the valley and about 40 ft. above the valley-bottom, a number of openings were made in the residual cherty clay, derived from the decay of the Knox dolomite, for manganese. Red, yellow, white, buff and purple-colored clays make up the residual cov-

ering in which the manganese openings were dug. Red clay is the surface-covering, and will average less than 5 ft. in thickness. The underlying yellow clay which predominates is highly siliceous and freely mixed with large and small chert-fragments, usually in an advanced stage of decay. The clays usually show partial stratification, the bedding-planes conforming in a general way with the ridge-slope. They thin toward the top of the ridge and are the thickest in the valley.

The ore is distributed through the clay in the form of stringers and masses only a few inches in thickness, which cut the clay in all directions, conforming at times with the bedding-planes. Many of the stringers are composed of quartz, around and along which the ore is deposited as impregnations, incrusta-

FIG. 13.



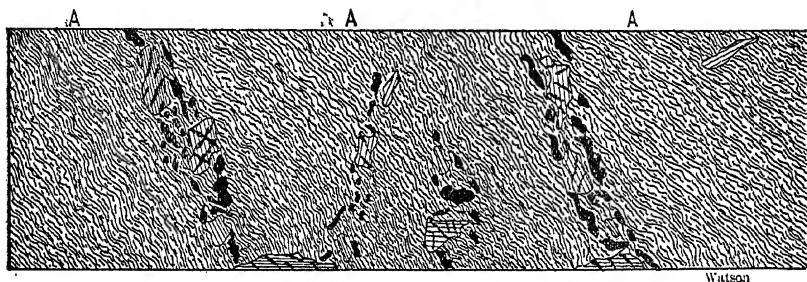
Cross-Section of Valley,  $\frac{3}{4}$  mile South of Lindale, Floyd Co., Georgia, Showing the Position of the Manganese-Deposit and the Relations of the Underlying Rocks.

A, residual clay, containing admixed chert and partially decayed rock-fragments; B, Knox dolomite; C, Conasauga shale. The black area is manganese.

tions, and as nodules and gravel. A goodly proportion of the gravel and nodular types of the ore is distributed through the clays without any apparent relation to the chert-fragments and masses. In most cases, however, the ore is closely associated with the chert, varying from impregnations as seams of knife-edge thickness to a ground-mass of ore cementing the partially fresh and decomposed chert-fragments. (See Fig. 15.) The form of breccia-ore commonly occurring in this locality is shown in Fig. 15. The proportion of chert to ore of the breccia-mass varies widely, from a mere film of manganese oxide, filling the cracks of the shattered chert and binding them together, to those in which the largest bulk of the mass is ore containing but few small chert-fragments.

*The Tunnel Hill District.*—The Tunnel Hill district includes the contiguous parts of Whitfield and Catoosa counties in the northeast part of the Paleozoic Group. In structure and topography the area quite closely resembles certain parts of the Cave Spring district already described. The rocks include shales, sandstones and limestones, and in age they range from Cambrian to Carboniferous. Several faults of the minor-thrust type cut the rocks at close intervals. The ores are limited to the residual decay derived from an underlying long, narrow belt of Knox dolomite, marking the position of a dissected chain of ridges trending northeast-southwest. This belt of the Knox formation is included between two faults, and

FIG. 14.



Section in One of the Openings at the Lindale Mine, 4 Miles South of Rome, Georgia, Showing the Mode of Occurrence of the Manganese-Ores.

A, ore-bearing clay. The black areas are manganese. The irregular areas with straight parallel lines are fragments of chert and sandstone, the cracks of which are filled with manganese oxide.

is in contact on the east and west sides with the Rome shales and sandstones.

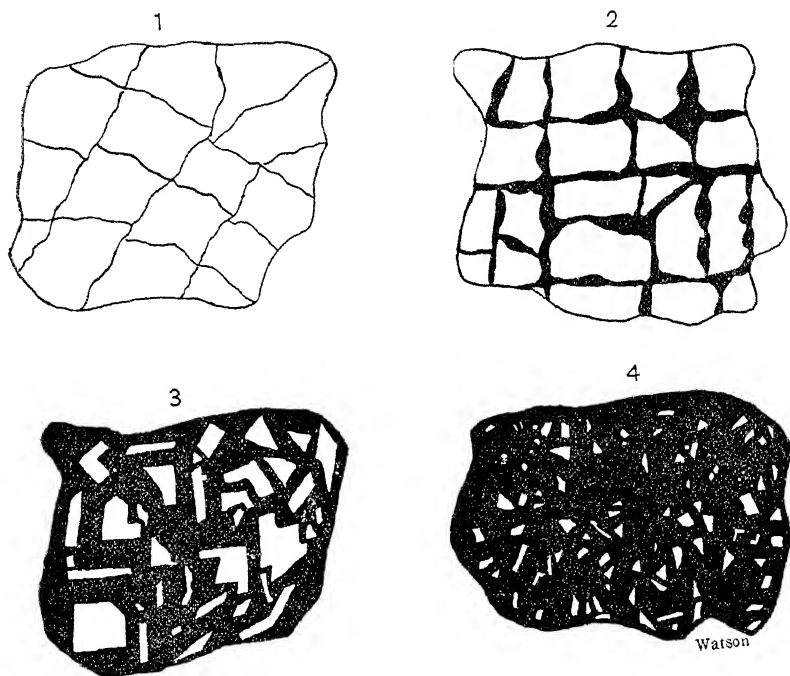
In 1890, the Catoosa Mining Company made extensive preparations for mining manganese on its property, located about 2 miles north of Tunnel Hill, a station on the Western and Atlantic railroad. A manganese-plant equipped with the necessary modern machinery was built, and a number of miles of railroad were laid from the mill to the openings. Less than 30 cars of ore, including manganese and manganiferous iron-ore, were shipped.

Openings were made along the magnesian limestone ridge for a distance of 6 miles northeast of Tunnel Hill. One shaft was sunk to a depth of 210 ft. in the residual cherty clays

without striking the bed-rock, and a number of others were put down to a depth of more than 100 ft. with the same result. A 150-ft. shaft is reported to have passed through manganiferous iron-ore for most of its depth.

The occurrence and character of the ore closely resemble that of the Cartersville district. The ore is mostly composed of botryoidal or kidney-shaped nodules, ranging from 1 to 12

FIG. 15.



Drawings Illustrating the Formation of Manganese-Breccia Ore in the Lindale and Cave Spring Deposits, Floyd Co., Georgia. Manganese oxide is represented by the black lines and areas. The white areas are fragments of chert and sandstone. Attention is directed to the increase in the proportion of manganese to rock in passing from No. 1 to No. 4.

and more in. in diameter, usually with a crystalline interior. The best exposures of the ore were observed in the cuts near the northern limits of the property.

The openings nearest Tunnel Hill show brown hematite and manganiferous iron-ore, much of which is of the breccia-type. The brown iron-ore is in close relation with the manganese, but is more abundant, occurring in the form of pockets and

lenticular layers as much as 20 ft. thick. The manganese and iron occur in many places admixed as manganiferous iron-ore; at other places they occur as separate and distinct ores in the same deposit; and at others still they occur as separate deposits without any admixture with each other.

*Origin of the Manganese-Ores in the Paleozoic Area.*

The stratigraphic position of the ores has been shown to be in the decay derived from and resting on three different formations belonging to the Cambro-Silurian, namely, the Weisner quartzite, Beaver limestone and Knox dolomite. The ores occur with about equal frequency in the decay derived from the three formations.

The character and depth of the decay and mode of occurrence, including distribution, of the ores in the clay have already been described in some detail, and need not be repeated here. While faulting is a characteristic structural feature of the region, the distribution, occurrence and nature of the ores preclude any connection or relationship to these lines of breakage. The region is, furthermore, one of extensive and closely associated ore-deposits of different mineral types, and each type is of considerable commercial importance. These have been separately and independently studied by different geologists in recent years, and while the deposits are closely associated, they have been shown to bear no genetic relationship to each other. It is necessary here to make clearer this association of deposits.

The more important associated deposits consist of brown iron-ore, yellow ocher and bauxite. The deposits of iron-ore all contain traces of manganese, and most of the manganese contains traces of iron, but the principal deposits of the two metals are quite distinct from each other. According to origin several distinct types of limonite or brown iron-ore occur in association with the manganese in the Paleozoic area, grouped by Hayes\* as (1) Gossan-ores; (2) Tertiary gravel-ores; (3) Concentration-deposits; and (4) Fault-deposits. Other forms of iron-ores occur, but they are of less importance within the immediate manganese districts. Only the

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\* *Trans.*, xxx., p. 411 (1901).



concentration- and fault-deposits concern the discussion of the genesis of the manganese accumulation.

Hayes refers to the concentration-deposits of the Cartersville district as follows:\*

“At various times these valleys [limestone] have received the drainage not only from the adjacent quartzite and limestone, but probably, also, from other of the valley formations; and the widely disseminated iron leached from these formations during the process of decay has been transported to the limestone valley, and there concentrated upon the underlying impervious quartzite.”

The principle underlying the genesis of the concentration-deposits is well expressed in the following sentence by Hayes:†  
“They may occur wherever a limestone is underlain by an insoluble and impervious stratum, such as sandstone or quartzite.”  
He further says:‡

“Favorable conditions for this accumulation occur in northwest Georgia and Alabama, at the contact of the lower Carboniferous limestones with sandstones which sometimes underlie it, and at the contact of the Beaver limestone with the underlying Weisner quartzite.”

The second type of iron ore-deposits of the area is that genetically related to the faults which intersect the strata and are designated by Hayes as fault-deposits.

The yellow ocher-deposits are closely associated with those of manganese in the Cartersville district; and also with the iron-ores of the same area. Beginning at a distance of about 3 miles southeast of Cartersville, the ocher-belt has a northward extension of 7 to 8 miles, confined exclusively to the Weisner quartzite, which has been greatly fractured and faulted. The occurrence of the ocher is entirely in the nature of replacement-deposits in the shattered quartzite,—the silica of the quartzite having been removed in solution and the hydrated ferric oxide substituted.§

The last type of ore-deposit in the area that needs mention is that carrying bauxite. The ore-bodies are distinct pocket-deposits, having the vertical and lateral dimensions about equal, inclosed in the residual clays derived from the decay of the

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\* *Ibid.*, p. 412-413.

† *Ibid.*, p. 412.

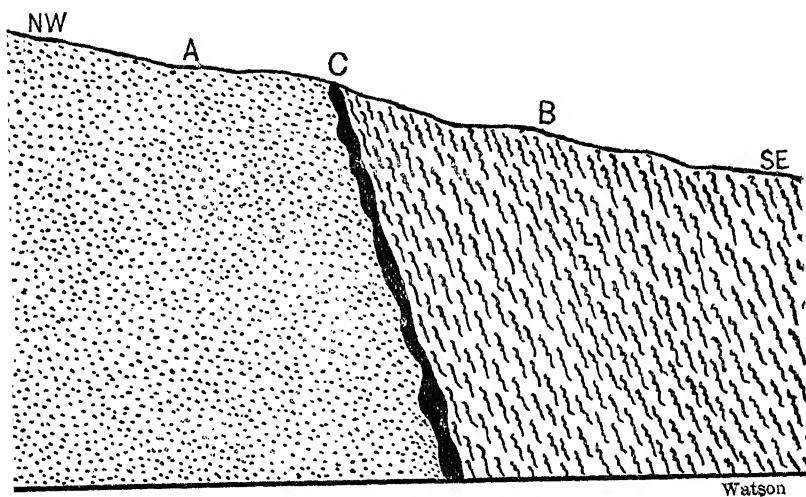
‡ *Ibid.*, p. 412.

§ *Ibid.*, pp. 415-418.

Knox dolomite. They were first shown by Hayes,\* whose views were afterwards confirmed by the writer,† to represent accumulations of hydrated aluminum oxide in vents or springs along the lines of numerous faults which intersect the area. The source of the alumina was from below, in the underlying aluminous shales of the Conasauga (Cambrian) formation, and was taken in solution by hot ascending acidulated waters circulating along the lines of fracture.

Enough detail has been given to show the non-relationship genetically between the manganese and the other ore-deposits.

FIG. 16.



Section through the Stratham Tract, near Draketown, Georgia, Showing the  
Mode of Occurrence of the Ores.

A, banded quartzite with magnetite and some pyrite; B, decayed mica-schist, in which the planes of schistosity are perfectly preserved; C, ore, including manganese and manganiferous iron-ore, magnetite and limonite.

We must look, therefore, to an independent theory for the genesis of the manganese. The theory which best accords with the facts as the writer has interpreted them is, with some modification, that essentially outlined previously by Penrose.‡

\* Hayes, C. W., *Sixteenth Annual Report U. S. Geol. Survey*, 1895, Part III., pp. 587-591.

† Watson, Thomas L., *American Geologist*, 1901, vol. xxviii., pp. 38-45.

‡ Penrose, R. A. F., "Manganese: Its Uses, Ores and Deposits," *Annual Report Geological Survey of Arkansas*, 1890, vol. i., p. 539 *et seq.*

It satisfactorily explains (1) the source from which the manganese was derived; (2) the method of solution, transportation and precipitation of the manganese; and (3) the process of its local accumulation.

1. *Source of the Manganese.*—The immediate source of the manganese was from the rocks from which the residual decay inclosing the ores was derived by weathering. Accumulation was not entirely limited, perhaps, to the manganese contained in any single formation in whose residual decay the ores now exist, but was derived from any formations containing this element, formerly covering the one in which the ores are now found. There is field-evidence in certain areas to support this statement.

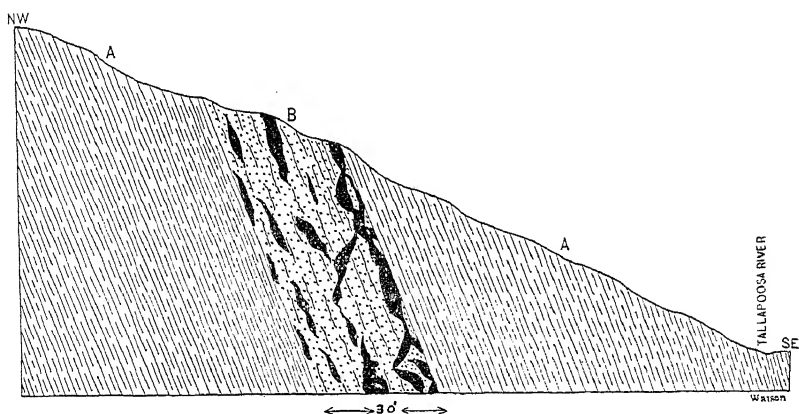
As shown from the general geologic distribution and mode of occurrence of the ores discussed above, the source of the manganese could not have been from rocks underlying those in whose residual decay the ores are now inclosed.

Microscopic study of a number of thin sections prepared from hand-specimens of the quartzite, collected by the writer from all parts of the quartzite area, failed to disclose any mineral substance that could be definitely referred to manganese in any mineralogical form. In order to further test the absence or presence of manganese in the quartzite, large fragments were chipped from each hand-specimen, mixed and powdered as one sample. From this bulk of powdered quartzite, a sample was carefully taken and subjected to a chemical analysis, searching particularly for manganese. The analysis showed no manganese (see analysis on page 215), which confirms the microscopic study. It would not be safe to conclude from a single analysis, though made from a sample prepared from hand-specimens of the rock taken over all parts of the area, that manganese was entirely absent from the formation. When added, however, to similar results from microscopic study, the two greatly strengthen such an inference. If these results should later prove conclusive, the source of the manganese in the decay of the quartzite must then have been from once-existing overlying formations, from which, upon weathering, the manganese was concentrated by chemical and physical conditions obtaining; or, else, the manganese now found in the decay of this formation was originally limited to that part of the formation reduced to decay.

The Paleozoic rocks are bordered on the east and south in Georgia by older crystalline rocks from which the former rocks were largely derived. These crystalline rocks, made up in part of original igneous masses and in part of original sedimentaries, are composed of numerous complex manganese-bearing silicates. It was from these older crystalline rocks, during decay, that the manganese is believed to have been originally derived.

2. *Solution, Transportation and Precipitation of the Manganese.*—Assuming that the original source of the manganese was from

FIG. 17.



Section through the Westbrook Tract, Paulding Co., Georgia, Showing the Mode of Occurrence of the Ores.

A, mica-schist, partially decayed, highly schistose; B, banded quartzite, 30 ft. wide, cut by quartz-stringers containing the ore, which includes manganese, manganese-iron-ore and magnetite.

the older crystalline rocks to the east and south, it remains to show how the manganese material reached its present form and position.

The crystalline rocks of Georgia are composed chiefly of granites, gneisses, schists and basic igneous masses, and they are everywhere deeply decayed,—buried under a thick covering of their residual clays. The essential minerals in these rocks are silicates, many of which are manganese-bearing. Decay in this southern region has been promoted largely by chemical changes in the mineral constituents of the rocks, resulting in mineralogical combinations of simpler and more

stable form, totally different from the original forms. The combined action of atmospheric oxygen, water, carbonic and organic acids, and to a less degree, perhaps, certain inorganic acids, has been the principal agent involved in the chemical decay of the rocks. Accompanying such changes, the metallic bases of the silicates combine with the various acids and are removed in solution as salts of these acids, the insoluble parts of the minerals remaining where formed, to make up the residual mantle. Manganese, with other of the base-forming elements, is thus removed in solution by the streams, and under favorable conditions of oxidation finally precipitated with the sediments on the floor of the water-bodies into which the streams drain.

Definite evidence from field-study in this area, by the writer, is lacking to indicate the exact form in which the manganese was laid down in the rocks, whether as carbonate or oxide or both.

After examination of the chemical behavior of manganese, Dunnington\* calls attention to the probability of manganese-sulphate having taken an important part in the formation of deposits of manganese-ores. In view of the results from this examination he says :†

“ It appears possible that many deposits of manganese in calciferous rocks owe their formation to the action of solutions of sulphates, and possibly an illustration of such action is presented in the manganese-deposits of Crimora, Augusta county, Virginia. . . . ”

Professor Dunnington then outlines the conditions under which he conceives the Virginia deposits to have been formed.

3. *Local Accumulation of the Manganese.*—If, as indicated, the manganese was regularly or irregularly disseminated in a finely divided state through the limestones and quartzite in greater or less quantity, then some secondary action or process must explain their present local accumulation. Segregation to any appreciable extent, if at all, of the finely disseminated particles of manganese does not appear to have taken place in the original unweathered rock. The agencies which promoted

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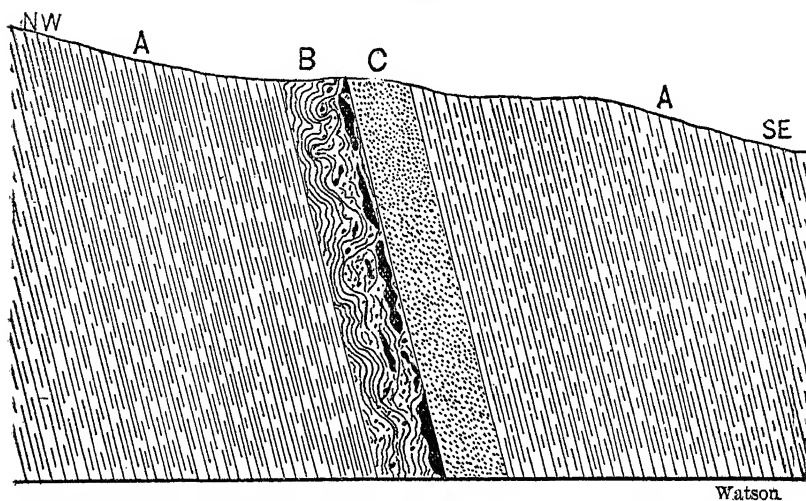
\* Dunnington, F. P., “On the Formation of Deposits of Oxides of Manganese,” *Amer. Journ. Sci.*, 1888 (3 s.), vol. xxxvi., pp. 175–178.

† *Ibid.*, p. 177.

the decay of the rocks inclosing the manganese particles were those involved in the accumulation of the ores in their present concentrated form. The process involved in the local accumulation was largely one of resolution of the manganese by the acidulated surface-waters and its reprecipitation in another position in the residual clays.

The irregular distribution of the ores, both laterally and vertically, in the residual clays; the frequency with which the ore-bodies are observed to cut across the bedding of the inclos-

FIG. 18.



Section through the Blue Ridge Mine, Fannin Co., Georgia, Showing the Mode of Occurrence of the Manganese- and Iron-Ores.

A, partially decayed mica-schist, grading into quartz-schist in places; B, manganese and iron oxides, distributed through the decay of the mica-schist. The black dots and areas are the ores; C, jasper-like quartz.

ing clays, and without regard to orientation in any direction; the invariable presence of greater or less quantities of included quartz-grains and other particles of siliceous material irregularly distributed through the nodules and masses of ore, of the same character as that composing the inclosing clays; the concretionary nodular and stalactitic forms of the ore; and its prevailing tendency to crystalline structure, are the most pronounced features of the ores, and are those which would result from such a process of segregation as outlined. That is, they are secondary accumulations, resulting from chemical and

physical action during the decay of the rocks containing the manganese.

Abundant masses of breccia-ore are associated to some extent with other types of the ore in all the deposits, but they are especially characteristic of the lower zone of decay of the quartzite, which consists of only partially-decayed and broken masses of the rock over the quartzite area. The formation of the breccia-masses is due to the downward percolation of the mangauiferous solutions through the overlying mantle of decay into the cracks and crevices separating the rock-fragments and deposition of the manganese oxide. In the Cave Spring district, where broken masses and beds of chert from the Knox dolomite abound, a similar formation of chert-breccia is observed. In some instances the percolation from above has extended into the cracks of the moderately fresh rock below, with deposition of manganese forming intersecting veins in the rock.

The frequent black color of the ore-bearing clays especially noticeable near the ore-bodies, due to the presence of very finely-disseminated particles of manganese oxide, finds explanation in the precipitation of the manganese oxide from the permeating solutions. All of these associations and different types of the ore are regarded as products of secondary chemical and physical action.

Finally, this process of local concentration of manganese has its analogy in the present accumulation of manganese in the residual decay of the crystalline rocks throughout the Southern Appalachians. The writer has observed in his field-study of rock-weathering, in parts of Virginia, the Carolinas and Georgia, that, in the weathered materials of these rocks, some of whose minerals were manganese-bearing, the decay was colored black in spots from the oxide of manganese, and, frequently, knife-edge stringers of the manganese were found filling the cracks in the clays.

The theory as outlined above is not new, but was previously elaborated in greater detail by Prof. Penrose in his exhaustive and excellent volume on manganese.\* The only point of difference between Prof. Penrose and myself, as to the Georgia

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\* *Annual Report Geol. Survey of Arkansas*, 1890, vol. i., p. 539 et seq.

ores, is that of the occurrence of some of the ore-bodies in the residual clays occupying similar positions in the original fresh rock, as stated by Prof. Penrose. The writer has not observed a single occurrence of the ores in the fresh rock in the Georgia area.

## II. MANGANESE-DEPOSITS OF THE CRYSTALLINE AREA.

The position of the Crystalline area is shown on the accompanying map (Fig. 1). The area includes two physiographically distinct provinces, namely, the Appalachian mountains and the Piedmont plateau. The transition in the rocks of the plateau, along its northwest margin, to those of the mountain province

FIG. 19.



Section along the South Face of the Large Opening on the Lowe Tract, near Cave Spring, Floyd Co., Georgia, Showing the Occurrence of Manganese "Pellet"-Ore in the Residual Clay.

Black dots indicate the "pellet"-ore; white areas, deep red-brown clay derived from the Knox dolomite. No admixed chert-fragments or nodular ore are contained in the clay at this point.

is indistinctly marked and is not sudden, but is, usually, gradual. Topographically, the exact limit between the two is equally difficult to define, since the elevation of the plateau near the border of the Appalachian mountain province is not sharply contrasted with that of the southeast margin of the latter province, but the slope of the one gradually passes into that of the other.

The Crystalline area forms the middle belt of the State (Fig. 1). It is separated on the southeast from the Coastal Plain by the fall-line, and on the northwest it is separated from the Paleozoic area by the Cartersville fault. Its axis has a general northeast-southwest trend, and, with the exception of the extreme ten northwest counties, it occupies the entire north part of the State.



*Rocks of the Crystalline Area.*

With but few exceptions, the rocks of the Crystalline area include profoundly altered, original clastic and igneous masses—crystalline-metamorphic rocks. Some of the granites and most, if not all, of the more recent basic dike-rocks retain in the field their original characteristic massive structures.

Many different mineralogical types of rocks are represented. Metamorphism has been so complete in many that it is often impossible to say, with certainty, whether they were derived from original sedimentary or igneous masses. Granites, gneisses, schists, slates, limestones and quartzites or sandstones compose the principal rocks. These are cut by numerous intrusions of more recent basic igneous rocks in the form of dikes. Diabase, diorite and gabbro comprise the commonest types of dike-rocks. Hornblende- and mica-schists are the most widespread of the crystalline schists. The granites and a part of the gneisses are mica-rocks.

The entire Crystalline area is of great complexity. The rocks are everywhere altered, intricately folded and tilted, and secondary structures have been induced in them. A further result of the intense metamorphism is the formation of many secondary minerals. The structural and age relations of the rocks of the area have not yet been worked out. With but few exceptions the rocks are geologically old, and belong to different periods of formation; some are pre-Cambrian, while others are of later age.

*The Rock-Forming Minerals of the Area.*

The rocks of the area comprise most of the more common rock-forming minerals and many of the rarer ones. The source of the manganese in the crystalline rocks of the State is chiefly from the various silicates containing manganese as one of the base-forming elements. Upon decomposition of the complex silicates, the manganese is either removed in solution in the form of a soluble salt and deposited with the sediments formed elsewhere, or it is retained in part or in whole in the form of the insoluble oxide distributed through the residual decay of the original rock, *in situ*. Numerous manganiferous silicates are distributed through the rocks of the Georgia Crystalline area, among the commonest of which are certain species belonging

to the amphibole, pyroxene, mica, garnet, epidote and olivine groups. Besides the silicate form of manganese, the writer has observed both the carbonate and oxide of manganese in several localities in the crystalline rocks of Georgia.

Rhodochrosite is found in Towns county, 2 miles west of Hiawassee, associated with the oxides of manganese and iron in hornblendic rocks of the corundum belt. Manganese oxide, in association with small grains and crystals of magnetite, occurs in a magnetite-quartzite schist in Haralson and Paulding counties.

### *Residual Decay of the Crystalline Rocks.*

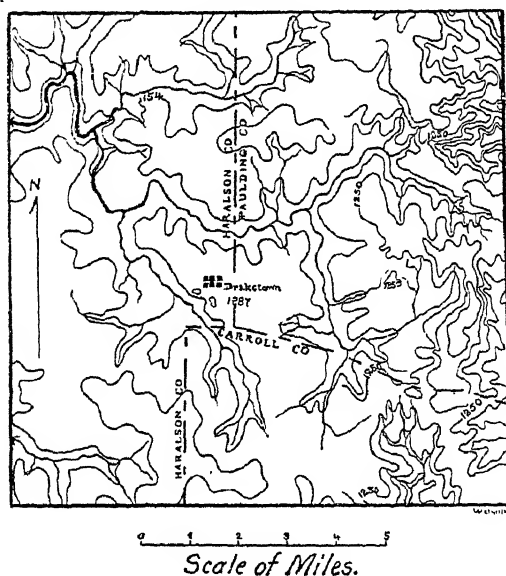
Atmospheric forces have been continuously operative on the rocks of the Crystalline area of Georgia for an indefinite period of time, resulting in the fresh rock being buried at present under a considerable depth of residual decay. Consequently, exposures of the fresh rock from which the decay was derived are seldom seen except on the steeper slopes and along the stream-courses. The mantle of decayed rock varies greatly in thickness, from a few feet to several hundred feet. Its character is equally variable, dependent mainly upon the type of rock from which it was derived and the forces promoting it.

### *Mode of Occurrence of the Manganese-Ores.*

The lithological associations and mode of occurrence of the manganese-ores are different from those of the Paleozoic area. The ore is usually massive and often fine-granular, admixed or otherwise closely associated with iron, and is less often of the gravel and larger concretionary nodular types so characteristic of the Cartersville district and other of the Paleozoic-area deposits. Of the former occurrence the locality to the northeast of Cohutta Springs, in Murray county, and that of the Drake-town district, in Haralson and Paulding counties, are perhaps the most typical. Near the Tennessee line, in the northeast part of Murray county and 5 to 6 miles northeast of Cohutta Springs, the manganese occurs as small nests or pockets in extensive beds of iron-ore in the Ocoee (pre-Cambrian) quartzites and slates. The manganese is not entirely free from iron, and much of it is a manganiferous iron-ore of apparently homogeneous composition.

The ore of the Draketown district in Haralson and Paulding counties is either massive or minutely-divided manganese oxide, in a finely-banded quartzite or sandstone intercalated with mica-schist. The siliceous rock is heavily charged with small grains and crystals of mangiferous magnetite and separate grains of manganese oxide, and in places the rock is pyritiferous. The manganese is mostly concentrated along the contact between the quartzite and schist, but contained mostly in the quartzite as massive ore carrying usually much

FIG. 20.



Topographical Map of the Draketown District, Georgia. (Tallapoosa and Marietta Topographic Sheets, *U. S. Geol. Survey.*) Contour-interval, 100 ft.

iron, which at times almost totally replaces the manganese. (See Figs. 16 and 17.) Between the two extremes of a manganese-ore containing a little iron and an iron-ore with a little manganese, all degrees of admixture of the two oxides occur.

The nodular type of ore similar to that of the Cartersville district is, perhaps, best developed in the manganese-deposits occurring within the southern limits of the town of Blue Ridge, in Fannin county. (See Fig. 18.) Here the manganese is found in the residual clays derived from the decay of mica-schist near the margin of a narrow band of jasper-like quartz,

which cuts the schist in an approximate north-south direction. The manganese is distributed through the clays as gravel, nodules and larger masses in nests or small pockets and stringers. The ore contains much siliceous impurity, and is in intimate relation with iron-ore, much of which has been shipped.

At other localities in the Crystalline area manganese is found, in massive and nodular forms and as a black clayey mixture of finely divided manganese oxide, in residual clays derived from the decay of hornblende- and mica-schists. The mica-schist is often garnetiferous.

The above occurrences of manganese in the crystalline rocks indicate (1) a concentration of the ore along and near the contact between certain formations; and (2) accumulation of the ore in the residual clays derived from the decay of various siliceous crystalline rocks.

*Mineralogical Forms of the Ores.*—The silicate, oxide and carbonate of manganese are found to some extent in the rocks of the Crystalline area. Neither the silicate nor the carbonate is of commercial importance. Manganese-bearing silicates are widely distributed among the more common rock-forming minerals of the area, and are of great importance in that they form the source, on decay, of the workable deposits of manganese in the form of the oxide. In several localities the oxide is found, in place, in the fresh rock, but, usually, it is a secondary product inclosed in the residual clays, similar to the ores of the Paleozoic area. The manganese of both the Crystalline and Paleozoic areas of Georgia include only the oxides of the metal.

*Extent of the Work and Location of the Deposits.*—More or less prospecting work for manganese-ores has been done in a number of counties in the Crystalline area. The test-work has been sufficient in most cases to indicate that no workable deposits of the ore exist, although small shipments of the ore have been made from a number of the openings in different counties.

The localities in which prospecting work has been done for manganese are widely separated and are scattered over various parts of the area which bear no apparent geological relationship to each other. The deposits are not associated with

any particular type of rock, but are found in association with several widely different mineralogical types.

The counties in which manganese has been worked or tested in the Crystalline area are Murray, Fannin, Towns, Cherokee, Haralson, Paulding, Habersham and Hart. See map (Fig. 1), which shows the distribution of the ore-deposits.

### *Genesis of the Ores in the Crystalline Area.*

Numerous openings made for manganese in different parts of the Crystalline area afforded opportunity for tracing the formation of the manganese oxides from several of the manganese-bearing silicates. Of these a manganiferous garnet and mica showed the formation of the oxide from the original silicate in both a partial and complete stage of decomposition. In each case the early stage was indicated by the original mineral being irregularly coated and spotted from decay by a mixture of the oxides of manganese and iron. The final stage showed the almost complete destruction of the original mineral, and its place filled by the black amorphous oxides of manganese and iron. More or less stain from the oxides had extended beyond the limits of the original manganese-bearing silicate, discoloring the inclosing clays.

The source of the manganese in the Crystalline area has been mostly, if not entirely, from the various manganese-bearing silicates, which enter largely into the composition of most of the rocks. By the decomposition of the minerals composing these rocks, the base-forming elements either form soluble salts of inorganic and organic acids promoting the decay, and are removed in solution, or else the whole or a part of certain of the base-elements are converted into insoluble oxides and retained, *in situ*, to form the mantle of residual decay. In either case the process may result in the removal by solution of only a part of certain of the base-elements, iron and manganese, in the form of soluble salts, while the remainder of the same elements is retained as insoluble oxides in the residual decay. Both reactions are common in the surface-zone of oxidation, as proved by recent work in rock-weathering.

Recent investigations in rock-weathering show that iron is frequently retained in the residual clays in amounts larger than that of any other constituent in proportion to the percentage

amount present in the fresh rock. A loss, however, by removal in solution, in the form of a soluble salt on decay of the rock, is often shown in the iron. So far as investigation has gone, the iron is not entirely removed in any case, but a part of it remains in the form of the insoluble oxide. A like tendency is indicated for manganese when present in those rocks so far investigated.

Upon subsequent chemical and physical changes, the manganese is further concentrated in the residual clays, and the accumulation of the oxide is sometimes in quantity sufficiently large to be of commercial value.

Some of the manganese-deposits of the Crystalline area indicate that the ore has been leached from the surrounding rocks upon decay and concentrated in the clays along the contact-zones of certain formations. In only one locality in the Crystalline area, namely, the Draketown district of Haralson and Paulding counties, has manganese been found in the form of the oxide, in place, in the original rocks. The manganese exists here partly as the free oxide in a banded quartzite, and partly as a manganese-bearing magnetite which forms a considerable percentage of the rock in places. Concentration of the manganese with iron has taken place near the margin of the quartzite in contact with mica-schist, in quantity sufficient to yield a small amount of workable ore.

Accompanying the process of rock-decay the retention of manganese in more localized form has been promoted in places. The process is still in progress, and, accompanying it, the accumulation of manganese in those places where the conditions are favorable.

Briefly stated, then, the manganese-ores of the Georgia Crystalline area represent the secondary accumulations of the insoluble oxides of the metal supplied from the manganese-bearing silicates on decomposition, and subsequently concentrated and localized in the residual clays derived from the decay of the underlying siliceous crystalline rocks. In places, accumulation of the oxides has progressed along and near contact-zones in the rocks; in other places concentration has been in the clays and removed from contacts.

*Methods of Mining the Ore.*

The nature of the ore to be mined in the Georgia area is one of irregular distribution, in the form of nodules and pockets, through residual clays, which range in thickness from 25 ft. to several hundred feet. The ore-distribution varies greatly, and the deposits are limited both in depth and lateral extent; hence, the methods for operating in one place will necessarily vary, more or less, in detail from those in another. As a rule, the deposits are located on the summits and higher slopes of the hills and ridges, though there are many exceptions, for they not infrequently occupy the lower slopes and valley-bottoms.

The method of mining will depend largely upon the location of the deposits and their depth below the surface. Open pit and cut, shaft and tunnel work are employed. These are often used together, to advantage, in the same place, especially where the ores begin at or near the surface and continue irregularly to some depth below. In such cases, open pit and cut work is used, and, from the bottom of the open work, shafts are sunk and drifts are run at different levels from the shafts. Tunneling becomes necessary in most of the steeper slope-deposits. In the lower deposits, especially those of the valley-bottoms, shafting and tunneling is most advantageously employed. In most cases of tunneling and shafting it becomes necessary, from the nature of the clays, to timber the openings in order to prevent caving. The timbering over most of the Georgia area has been poorly done, and, in many cases, put in to meet only temporary needs.

Expensive and heavy machinery is unnecessary, and the equipment should be as light and portable as possible, so that moving from one place to another, as the ore becomes exhausted, can be done speedily and at a small cost.

*Preparation of the Ore.*

The occurrence of the ores in the residual clays means, usually, more or less admixture of the ore with clay. Usually, the only treatment of the ore necessary before shipping is to free it from the adhering clay. Crushing and jigging are necessary in the spongy or porous type of ore, the numerous cavities of which are filled with the clay; also in those ores containing

considerable free-quartz grains and cemented fragments of the rock. This is especially true of much of the breccia-ore, which is rendered marketable by materially reducing the amount of siliceous material in this way. Washing will usually suffice for cleansing the bulk of the ore. In the crushed and jigged ore, subsequent washing is also necessary.

In the early history of manganese-mining in Georgia less care was used in properly cleansing the ore than at present, and much of the ore then shipped contained large quantities of adhering clay and other extraneous material. The principal and, frequently, the only treatment was that of screening. At that time all the washing done was by hand. The form of washer used was a revolving-cylinder perforated with holes, and fed inside by a constant stream of water. The ore was put into the cylinder through a door, the door closed and the cylinder revolved by hand until the ore was freed by running water from the clay, when the ore was removed through the same opening. The capacity of the washer was very limited, and it could be used only on a small scale. Much of the smaller ore was lost, but the larger fragments were thoroughly cleansed.

Later, a form of log-washer, similar to that used for cleansing the brown iron-ores, was introduced, and is the form of washer at present in use. Briefly, the log-washer consists of a long and stoutly-built box, of sufficient length and depth to contain the log. The box, or trough, is elevated at one end. A log or central shaft, from 25 to 40 ft. long, carrying heavy iron-flanges, spirally arranged the length of the log, revolves lengthwise in the box or trough. The ore is fed at the upper end of the box, and is gradually forced by the revolving log to the lower end, where it passes out. A constant stream of water plays on the ore in the box. The constant agitation and beating of the ore by the log and the washing by water frees it from the adhering clay. Steam is the motive-power.



## Notes on the New Jersey Fire-Brick Industry.\*

BY HEINRICH RIES, ITHACA, N. Y.

(Albany Meeting, February, 1903.)

THE manufacture of fire-brick represents one of the oldest branches of the clay-working industry in New Jersey, and is of more importance than is commonly imagined.

The New Jersey clays were first used for fire-brick after the war of 1812, and one of the earliest records, according to Professor Cook,† shows that clay was taken from Woodbridge to Boston in 1816, and used for that purpose.

The value of the clays of the Woodbridge district does not seem to have been well and widely recognized for some years, however, although in 1855 the statistics, given in the Geological Report, show that clay for making 50,000,000 fire-bricks was then taken annually from the pits at Woodbridge, Perth Amboy and South Amboy.

Perhaps the oldest works in the State was that known as the Salamander works (no longer in existence), where brick were made as early as 1825. A little later, in 1836, John R. Watson established a factory at Perth Amboy, and in 1858 Sayre & Fisher began the manufacture of fire-brick at Washington.

The works of W. H. Berry at Woodbridge began operations in 1845, and have continued up to the present day, although in 1896 the name was changed to J. E. Berry.

The works of Henry Maurer & Son were established in 1856, and those of M. D. Valentine & Brother in 1865. These latter were started for making bath brick; later, sewer-pipe, and, finally, fire-brick and other refractory shapes were produced. A branch-works, located at Valentine on the Central railroad of New Jersey, was started in 1887.

Even in the last two years several works have begun the manufacture of fire-brick. These include factories erected by

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\* Published by permission of the State Geologist of New Jersey.

† "Report on Clays," *N. J. Geol. Survey*, p. 1, 1878.

the Mutton Hollow Fire-Brick Co., Anness & Potter, at Woodbridge, and The Superior Fire-Lining Co., at Trenton.

In addition to these several other firms have made fire-brick from time to time, but they do not represent a regular item of production. The industry has thus grown, so that in 1901 the value of the fire-brick produced in New Jersey was \$780,327, the State being outranked in this matter only by Ohio and Pennsylvania.

The fire-brick produced in New Jersey are made from a mixture of fire-clays dug around Woodbridge or other points in eastern and southeastern Middlesex county, and to this there is added a certain percentage of ground fire-brick (grog), some siliceous clay and, occasionally, a gravelly feldspar.

Table I. shows the character of some of the fire-clays:

TABLE I.—*Analyses of New Jersey Fire-Clays.*

	1	2	3
Silica ( $\text{SiO}_2$ ), . . . . .	40.64	46.90	71.80
Alumina ( $\text{Al}_2\text{O}_3$ ), . . . . .	41.19	35.90	18.92
Ferric oxide ( $\text{Fe}_2\text{O}_3$ ), . . . . .	3.27	1.10	0.88
Lime ( $\text{CaO}$ ), . . . . .	0.65	.....	.....
Magnesia ( $\text{MgO}$ ), . . . . .	trace	.....	.....
Alkalies, . . . . .	trace	0.44	0.48
Water, . . . . .	14.74	12.80	6.70
Moisture, . . . . .	.....	1.50	0.50
Titanic acid ( $\text{TiO}_2$ ), . . . . .	.....	1.30	.....
	100.49	99.94	99.28

1. Blue clay, Crossman's pits. L. C. Graton, analyst.

2. No. 1 fire-clay. Woodbridge.\*

3. Extra sandy clay. Woodbridge.\*

The refractoriness of the clays, when tested in the Deville furnace, ranges commonly from cone 33 to cone 35 for the No. 1 fire-clays, and from cone 27 to cone 30 for the No. 2 fire-clays.

Without going into the details of their physical characteristics, which are given in full in a forthcoming report of the New Jersey Geological Survey, it may be sufficient to say that they are commonly of good plasticity, but low tensile strength, the latter ranging from 40 to 50 lb. per sq. in. Some of the No. 1 grades crack in burning, and therefore they can be used

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\* "Report on Clays," *N. J. Geol. Survey*, p. 297, 1878.

only to a limited extent in the fire-brick mixtures. The fire-shrinkage, when burned to cone 8, may range from 6 to 12 per cent., depending on the sandiness of the material.

The methods of manufacture used are those prevalent in most States where fire-brick are manufactured, the clay being tempered in ring-pits, and molded usually either by hand or in soft-mud machines. A few manufacturers employ the stiff-mud process. After molding, the bricks are re-pressed and burned with few exceptions in down-draft kilns. The temperature of burning varies slightly at the different works, but is generally between cones 10 and 12.

Many of the first grade of fire-clays found in the State are highly refractory, and it might be supposed that the bricks made from them would show an equal degree of refractoriness. This is not the case, however.

Table II. gives partial analyses of various New Jersey fire-bricks, with the cone-numbers at which they fuse.

TABLE II.—*Composition and Fusibility of New Jersey Fire-Bricks.*

	SiO <sub>2</sub> .	Al <sub>2</sub> O <sub>3</sub> .	TiO <sub>2</sub> .	Fe <sub>2</sub> O <sub>3</sub> .	CaO, MgO, Alkalies. By Differ- ence.	Cone of Fusion.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Number.
1.....	77.30	18.99	1.05	2.06	0.60	31-32
2.....	82.45	14.86	0.90	1.54	0.25	32
3.....	68.70	26.50	2.65	1.25	0.90	27
4.....	77.15	19.46	0.95	1.99	0.45	27
5.....	82.30	14.84	0.71	1.70	0.45	31
6.....	75.20	21.09	1.72	1.37	0.62	33
7.....	77.70	19.51	1.86	1.11	.....	28-29
8.....	79.84	18.06	1.53	1.65	.....	31

This series, which presents the range of composition found in New Jersey fire-bricks, brings out some interesting facts.

It will be noticed, in the first place, that the fusion-point ranges from cone 27 up to cone 33.

Looking over the analyses, we find that most of the bricks are quite siliceous, few of them containing less than 70 per cent. of silica. The alumina contents are not high, neither is the percentage of Fe<sub>2</sub>O<sub>3</sub> large. A noteworthy feature is the universally high percentage of TiO<sub>2</sub>.

The fact that the fusibility of these bricks lies at points between cones 27 and 33, instead of higher, is not due alone to fluxing-impurities, such as iron oxide and alkalis, but, doubtless, in a large degree to the ratio between silica to kaolinite-base in the clay. Few fire-brick manufacturers, even at the present day, seem to recognize the fact that silica at high temperatures act as a flux when heated with a certain proportion of alumina. It seems to the writer that in the case of the New Jersey fire-bricks we have an example of this disregard of a chemical fact.

This statement, however, involves no intimation that the New Jersey bricks are of poor quality, for many of them bear an excellent reputation, and often are found to stand frequent and great changes of temperature better than other bricks. They are used extensively in blast-furnaces, kilns, cupolas, etc., with good results.

The  $\text{TiO}_2$  in the clay acts as a stronger flux than silica, percentages of but 2 or 3 per cent. added to a high-grade kaolin, lowering its fusing point two or three cone-numbers.

Unfortunately, but little systematic work has been done in this country towards tracing out the relation between the refractoriness of fire-bricks and their composition.

An interesting series of experiments has recently been published in the *Thonindustrie-Zeitung*, vol. xxvii., p. 767 (1903), in which it is shown that the more siliceous bricks, even when low in fluxes, are less refractory. Table III., compiled from the article mentioned, gives examples:

TABLE III.—*Composition and Fusibility of Foreign Fire-Bricks.*

	$\text{SiO}_2$ .	$\text{Al}_2\text{O}_3$ .	$\text{Fe}_2\text{O}_3$ .	$\text{CaO, MgO, Alkalis.}$	Cone of Fusion.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Number.
Saarau, Germany.....	52.20	45.90	1.00	0.90	35-36
Saarau, Germany.....	69.65	28.71	1.20	0.44	below 33
Lucas, England.....	61.24	30.48	3.20	4.89	" 31
Stourbridge, England.....	63.40	31.70	3.00	1.90	30-31

In the first two it will be seen that there is little difference in the total of fluxes, but considerable difference in the silica-percentage; and that the first fuses fully three cone-numbers higher than the second.

## Notes on the Metallurgy of Copper of Montana.

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(Albany Meeting, February, 1903.)

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## I. INTRODUCTORY.

IN 1901,\* this country produced 268,782 long tons of fine copper. Of this total, Montana is credited with 102,621 long tons, or 38.2 per cent.

\* *United States Geological Survey*, "Mineral Resources," 1901, p. 160.

The leading smelting-works are located in Butte, Anaconda and Great Falls. In Butte are the works of the Colorado Smelting and Mining Co., the Colusa Smelter of the Montana Copper Co. (closed in 1893), the plant of the Parrot Silver and Copper Co. (closed in 1899), the old Bell plant (closed in 1884), the Clark's Colusa (closed in 1893), the active works of the Colusa Parrot Mining and Smelting Co. (Butte Reduction-Works), the Butte and Boston Consolidated Mining Co., and the Montana Ore-Purchasing Co.

In Anaconda are the old and new (Washoe) works of the Anaconda Copper Mining Co.; in Great Falls the establishment of the Boston and Montana Consolidated Copper and Silver Mining Co.

The active works furnished in 1901, approximately, the following amounts :

Anaconda Copper Mining Co., . . . .	113,800,000 lb.
Boston and Montana C. C. & S. M. Co., . . . .	54,700,000 "
Montana Ore-Purchasing Co., . . . .	29,900,000 "
Butte and Boston Con. Min. Co., . . . .	22,200,000 "
Butte Reduction-Works, . . . .	18,000,000 "
Colorado Smelting & Min. Co., . . . .	7,500,000 "
	<hr/>
	246,100,000 "

The discrepancy of 17,000,000 lb. of copper between the figures of production is due to the fact that the matte of some works is converted in others, and that both are credited with the copper they turned out.

## II. CONDENSED ACCOUNT OF PAST AND PRESENT PLANTS.

### *Colorado Smelting and Mining Co.*

This plant was started in 1879 by the late Henry Williams.\* The ore was roasted at first in the long-hearth hand reverberatory-furnace (hearth 12 by 50 ft.), and then smelted in a reverberatory matting-furnace (hearth 14 ft. 7 in. by 9 ft.), wood being used as fuel. The original smelting-furnaces treated 12 tons of ore in 24 h., producing matte assaying 60 per cent.

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\* R. W. Raymond, *Engineering and Mining Journal*, July 19, 1902; J. Douglas, *The Mineral Industry*, iv., 283; C. W. Goodale, *International Mining Congress*, Butte, 1902.

copper and from 700 to 800 oz. silver to the ton, which had to be hauled 200 miles to reach the railroad. In 1889 Brückner cylinders came into use for roasting ores; followed in 1892 by the R. Pearce single-deck turret-furnace. In 1894 R. Pearce's first double-deck turret-furnace was added, and in 1901 the first six-deck turret-furnace of R. F. Pearce was put in operation. At present (summer, 1902), the roasting department has 4 single-deck turret-furnaces with 6-ft. hearths, 5 double-deck turret-furnaces (1 with a 6-ft. and 4 with a 7-ft. hearth), and one 6-deck turret-furnace with a 7-ft. hearth.

The hearths of the reverberatory matting-furnaces were made 21 by 12 ft. in 1889, increasing the capacity to 18 tons in 24 h. At present, there are 3 matting-reverberatories with hearths 47 by 20 ft. (the first of these larger furnaces having been erected in 1899). The matte containing copper, from 50 to 55 per cent.; silver, 100 oz.; and gold, 1.5 oz. to the ton, is sold.

*Colusa Smelter of the Montana Copper Co.*

This smelter was started in 1880 by A. Wartenweiler, with 4 long hand-reverberatory roasters (hearths 60 ft. by 11 ft. 6 in.), and 2 reverberatory matting-furnaces (hearths 15 ft. 9 in. by 10 ft.). Smelting was begun in 1881; a reverberatory matting-furnace treated in 24 h. 12 tons of raw and roasted ores, producing 65-per cent. copper-matte, with a consumption of 10 cords of wood. Lump-ore was at first heap-roasted, but, on account of the smoke-nuisance, the heaps were replaced in 1882 by stalls.\* In 1884† there were in operation, besides the stalls, 12 long-hearth hand reverberatory-roasting, 6 reverberatory-matting, and 1 blast-furnace. The latter, 3 ft. by 4 ft. 6 in. at the tuyeres and 10 ft. high, was run with charcoal. The works were sold in 1888 to the Boston and Montana Con. Copper & Silver Min. Co., and formed the so-called Upper Works. They were closed down in 1893, when the new Great Falls establishment of the company made them superfluous.

*Parrot Silver and Copper Co.*

The works were started in July, 1881. Ore was at first roasted in 11 long hand-reverberatories (hearths 60 by 14 ft.)

\* *Mineral Resources of the United States*, 1891, p. 94.

† *Mineral Resources of the United States*, 1883-84, p. 339.

and smelted in 6 matting-reverberatories (hearth 14 ft. by 11 ft. 6 in.). The smelting-charge weighed from 2 to 3 tons; 10 tons were put through a furnace in 24 h. The matte (60 to 67 per cent. copper and 40 to 45 oz. silver per ton) was at first brought forward to blister-copper, but this was soon given up on account of the excessive cost, and the fact that matte was more easily marketed than blister-copper. Shortly after starting of works, a stall, 40 ft. deep and 9 ft. high,\* was built by J. T. Reese to roast lump-ore. It proved a failure and was replaced by A. J. Schumacher with open stalls,† 8 ft. deep, 6 ft. high and 6 ft. wide. In 1891 there were in operation 120 stalls.‡ The hand-reverberatory roasting-furnaces were supplemented in 1889 by 2 small Spence mechanical shelf-roasting furnaces,§ each treating 5 tons of concentrates in 24 h., reducing sulphur to from 6 to 7 per cent.; these were replaced in 1892 by a round, and in 1893 by 2 oblong Keller-Gaylord-Cole mechanical shelf-burners.||

A Roberts 4-hearth mechanical reverberatory-roaster, made by the Western Iron-Works of Butte, and similar to the Wethey furnace,¶ was put in operation in 1897. The hearths of the matting-reverberatories were enlarged in 1887 to 16 ft. by 12 ft. 6 in. For lack of space they never reached the large sizes of the other furnaces at Butte. The largest furnace, built in 1895, had a hearth of 22 ft. by 16 ft. In 1884 a Herreshoff 48-in. water-jacket blast-furnace was erected by E. D. Peters; in 1885 a 3- by 6-ft. blast-furnace, with cast-iron water-jackets, was added, but was soon replaced by A. J. Schumacher with a 3- by 8-ft. furnace having steel jackets. In 1884 the converting of copper-matte was begun under the general direction of the inventor of the process, Pierre Manhès. The practice was to melt 40- to 50-per cent. copper-matte in a cupola-furnace, tap it into converter, blow to white metal and pour, then melt the white metal in the cupola and blow it to blister-copper in the converter. In 1885 A. J. Schumacher succeeded in making

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\* Width, unknown.

† E. D. Peters, *Mineral Resources of the United States*, 1883-84, p. 389; also, *Modern Copper-Smelting*, 1895, p. 181.

‡ *Mineral Resources of the United States*, 1891, p. 98.

§ Peters, *Modern Copper-Smelting*, 1895, p. 220.

¶ Peters, *op. cit.*, p. 214; Hofman, *Metallurgy of Lead*, 1899, p. 191.

¶ Hofman, *op. cit.*, 1899, p. 195; *The Mineral Industry*, 1898, vi., 449.



metallic copper from 40- to 50-per cent. matte in one operation of two consecutive stages, and thus laid the foundation of the modern practice of converting.

When the works were closed down, August 20, 1899, the roasting department contained 2 Keller-Cole-Gaylord furnaces, each having a capacity of 50 tons ore, and 1 O'Hara furnace, treating 50 tons in 24 h. Smelting was carried on in 3 undergrate blast reverberatory furnaces with hearths 20 by 12 ft., treating altogether 135 tons of ore in 24 h., and in 2 blast-furnaces, 44 in. by 96 in. at the tuyeres, each putting through 120 tons of charge daily. The converting department had 6 upright converters, 5 ft. in diameter and 8 ft. 6 in. in total height.

#### *Bell Plant.*

Soon after the erection of the Parrot works followed the building of the Bell plant, which roasted in long-hearth reverberatories and smelted in blast-furnaces. In 1885\* it had 2 hand-roasters and 2 blast-furnaces.

#### *Clark's Colusa.*

These works were built in 1884 by W. A. Clark; were sold in 1888 to Boston and Montana Co., forming its Lower Works; and closed in 1893 in favor of the new plant at Great Falls. At this plant the first 2-hearth, mechanical O'Hara roasting-furnace was operated, 1885; the hearths were 50 by 7 ft. A second O'Hara furnace, erected in 1885, had hearths 70 by 7 ft., and treated 30 tons ore in 24 h. In 1891† ores were roasted in 3 O'Hara furnaces and one 10-ton Brückner cylinder. The first smelting-furnace, a circular, water-jacket blast-furnace, 36 in. in diam. at the tuyeres and 9 ft. high, was replaced in 1885 by a larger furnace. In 1887, two 50-ton circular, water-jacket blast-furnaces were put in operation. In 1886, two reverberatory matting-furnaces were added to the plant, each treating 12 tons of charge in 24 hours.

#### *Colusa-Parrot Mining and Smelting Company's Plant (Butte Reduction-Works).*

This was started in 1885 by employees of the Parrot Smelter. At first, ore was roasted in heaps and then smelted (45 tons in

\* *U. S. Geological Survey, Mineral Resources of the United States*, 1885, p. 216.

† *Id.*, 1891, p. 94.

24 h.) in a 36-in. water-jacket blast-furnace. A long hand-reverberatory (hearth 100 by 14 ft.), erected in 1886, remained in operation until 1888, when it was cut in two, making 2 furnaces, 50 by 14 ft. each. In the same year the first Brückner cylinder was installed (15 by 7 ft.; charge, 7 tons). In the fall of 1889, a second cylinder, 20 by 8 ft. was added. In 1894 the first 4-deck Wethey calciner was built, replacing one of the 50-by 14-ft. hand-reverberatory roasters and the 15- by 7-ft. Brückner cylinder. Its work proving satisfactory, a second Wethey furnace was erected. The 50- by 14-ft. hand-roaster was replaced by a larger furnace, 75 by 5 ft., to be used mainly for roasting slimes. In 1898 the third, and in 1900 the fourth Wethey furnace was installed, replacing the remaining hand-roaster and Brückner cylinder. The smelting capacity of the plant was increased in 1886 by the erection of a 96- by 42-in. Orford brick-furnace. The original 36-in. furnace was put out of blast and replaced in 1889 by one of 38 in. diam. at tuyeres. In 1897 a sectional blast-furnace of the United Verde Hot-Air type, 84 by 42 in. at tuyeres, replaced the two older furnaces. In 1900 a second rectangular furnace, 112 by 42 in. at tuyeres, was put in blast, and in 1901 the 84- by 42-in. furnace was enlarged to 112 by 42 in.

The first reverberatory matting-furnace (hearth 15 by 9 ft.) was erected in 1887; the second (hearth 18 by 11 ft.), treating from 12 to 15 tons in 24 h., in 1888. It was then customary to wet down roasted ore and shovel it into matting-furnace through side-doors. In 1889, hoppers were placed over the furnaces, and roasted ore was charged hot through the roof. This increased the smelting-capacity from 18 to 21 tons in 24 h. In 1896 a large reverberatory matting-furnace (hearth 35 by 16 ft.; capacity from 45 to 50 tons ore in 24 h.) replaced the two smaller furnaces. A second (hearth 45 by 16 ft.; capacity from 55 to 70 tons) was built in 1897; and in 1900 a third (hearth, 50 by 20 ft.; capacity from 85 to 100 tons). This last did so well that the 1896-furnace was rebuilt on the lines of the furnace of 1900. The present plant comprises 4 Wethey 4-deck calciners, 2 water-jacketed blast-furnaces (112 by 42 in. at tuyeres), and 3 reverberatory matting-furnaces (hearths 45 by 16 ft., 50 by 20 ft., and 50 by 20 ft.).

*Butte and Boston Consolidated Mining Co.*

These works were started in 1885 with 4 Brückner cylinders (8 by 18 ft.), 2 reverberatory matting-furnaces (hearths 22 by 17 ft., treating 5-ton charges), and 2 blast-furnaces, the larger being 96 by 36 in. at the tuyeres and 11 ft. high. In 1889 2 O'Hara roasting-furnaces were added; later, 4 Allen-Brown-O'Hara furnaces; and, these giving satisfaction, the original O'Haras were remodeled to the Allen-Brown-O'Hara type, and the Brückner cylinders thrown out. In 1892, when the plant burnt down, 8 Allen-Brown-O'Hara furnaces were in operation. The plant was immediately rebuilt, the Allen-Brown-O'Hara furnaces were repaired, but not the Brückner cylinder. A new blast-furnace, 96 by 36 in. and 16 ft. high, was erected to take the place of the old small cupola. This furnace was originally provided with a bell-and-hopper feed, which proved too expensive. In 1901 the width of the new furnace was increased to 40 in. and the height reduced to 11 ft. Up to 1893 the blast-furnaces smelted little but roasted concentrates. The matting reverberatory-furnaces were remodeled and enlarged in 1895. The converter-plant was started in 1900. The present plant comprises 8 Allen-Brown-O'Hara roasting-furnaces (hearths 90 by 9 ft.), 4 matting-reverberatories (hearths 50 by 20 ft., the first furnaces of this size having been built in 1899-90), 1 blast-furnace 96 by 44 in. at tuyeres and 9 ft. high, and 3 horizontal converters with barrels 10 ft. 6 in. by 7 ft.

*The Montana Ore-Purchasing Co.*

Ground was broken for smelting-works Oct. 17, 1892; first matte was tapped January 1, 1893. The ore was roasted in 2 Allen-Brown-O'Hara furnaces (hearths 9 by 100 ft., each treating 60 tons in 24 h.), and smelted in 2 reverberatory matting-furnaces (hearth 12 ft. 10 in. by 20 ft., and fire-box 5 by 6 ft; natural draught, stack 80 ft. high and 4 by 4 ft. in the clear, and charge 7 or 8 tons; time of melting 3.5 or 4 hours) and one blast-furnace, 42 in. diam. at tuyeres. After a year's run, the blast-furnace was replaced by 2 matting-reverberatories like those first built. In 1894, two Herreshoff furnaces (9 ft. inside diam., 5 hearths) were added to the roasting-department. These giving satisfaction, 8 more were erected at once, 30 in

1895, and 30 in 1899, making in all 70 Herreshoffs. The O'Haras were destroyed by fire in 1901, and not rebuilt. Early in 1894 the hearths of the matting-reverberatories were enlarged to 13 ft. 6 in. by 25 ft., fire-boxes, however, remaining 5 ft. by 6 ft.; at same time the fuel was changed from lump to slack coal, and undergrate-blast substituted for natural draught. These changes increased the tonnage per furnace for 24 h. by 10 or 15 tons. The furnaces were again enlarged in 1898, and 2 more were built in 1899, making 6 in all (hearth 16 by 33 ft., fire-box 6 by 8 ft., stack 80 ft. high and 5 by 5 ft. in the clear).

In January, 1900, 2 blast-furnaces (42 by 120 in. at tuyeres; 11 ft. from tuyeres to throat; cast-iron water-jackets) were blown in. These water-jackets giving much trouble by cracking, the furnaces were replaced by larger furnaces (42 by 160 in. at tuyeres and 10 ft. 6 in. in height) with steel water-jackets. The converting-plant was started in August, 1893, with 2 vessels of the Parrot type (5 ft. external diam.); in 1894 two more were added, and in 1899-1900 four more. Two barrel-converters (bowls 10 ft. 6 in. long and 7 ft. in diam.), installed in 1901, are in constant use, and the upright Parrot converters are worked only when these are unable to handle all the matte. It is the intention to erect 2 additional horizontal converters and to remove the remaining upright ones. The present plant comprises 70 Herreshoff roasters, 6 reverberatory matting-furnaces, 4 blast-furnaces, 2 horizontal converters and 8 upright ones.

#### *The Anaconda Copper-Mining Co.*

At Anaconda there are 3 separate plants, the Upper, the Lower, and the New or Washoe Works. The Upper Works of the Anaconda Copper-Mining Co. were started by Marcus Daly in 1883 with 34 hand-reverberatory roasting-furnaces (hearths 50 by 14 ft.), 26 reverberatory matting-furnaces (hearths 20 by 14 ft.), and two 70-ton water-jacket blast-furnaces. A reverberatory smelting-charge weighed 3.5 tons, and 15 tons were treated in a furnace in 24 hours. After the fire of 1889, the works were rebuilt, and remained unchanged until closed down in November, 1901. At that time they contained 40 Brückner roasting-furnaces with cylinders 18 by 9 ft., 4 MacDougall roasting-furnaces (16 ft. diam. and 21 ft. high, with 6 hearths), 4 Wethey calciners, with hearths 100 by 12 ft., and

11 reverberatory matting-furnaces, with hearths 31 by 16 ft., a matting-furnace taking 9-ton charges and treating 60 tons of ore in 24 h.

The Lower Works of the Anaconda Copper-Mining Co., located about 2 miles below the Upper Works, were built under Marcus Daly by O. Stallmann. The smelting-department was started in December, 1888, with 56 Brückner cylinders (9 ft. 6 in. by 18 ft.) and 28 reverberatory matting-furnaces, with hearths 22 by 16 ft., a matting-furnace receiving 6-ton charges and treating 40 tons of ore in 24 h. The smelter buildings, which were entirely of timber, were totally destroyed in March, 1889, by a fire which started in the coal-bins. They were quickly replaced by a steel structure. In 1890, 40 Brückner furnaces of the same size as the first 56 were added to the plant, and in 1899 were added 4 MacDougall furnaces of the Evans-Klepetko pattern (16 ft. diam. and 21 ft. high, with 6 hearths). Of the 28 original matting-furnaces, 7 were enlarged in 1899; 3 had hearths 32 by 18 ft., 33 by 18 ft. and 45 by 18 ft.; and 4, hearths 35 by 18 ft. They were charged with about 11 tons of ore and averaged 75 tons in 24 h.

The converter-plant was started in 1890 with 15 stalls for upright vessels, 10 ft. high and 6 ft. in diam. In the same building were 6 blast-furnaces, 8 ft. by 3 ft. 6 in. at tuyeres and 10 ft high.

The New Electrolytic Refining Plant,\* still in full operation, started in 1894, refines part of the converter-copper of the Washoe works, the rest being shipped to Baltimore for treatment. The old experimental plant,† started in 1891 and closed in 1893, contained 630 tanks.

The New or Washoe Works‡ of the Amalgamated Copper Co. were planned and erected by F. Klepetko. They are located on the site selected by the Washoe Copper Co. (incorporated about 1893) for a new plant. When the Amalgamated Copper Co. obtained control of the Anaconda and Washoe companies, it was decided to erect new works instead of remodeling and enlarging those of the Anaconda Copper Co.

\* *Berg und Hüttenmännische Zeitung*, 1893, p. 53. *Zeitschrift für Electrochemie*, 1901, vol. vii., p. 793. *Engineering and Mining Journal*, Sept. 19, 1896, p. 271.

† *Mineral Resources of the United States*, 1891, p. 92.

‡ *Engineering and Mining Journal*, March 1, 1902, p. 311. *Western Mining World*, March 8, 1902.

The works were started in 1902, contain 48 6-hearth MacDougall-Evans-Klepetko roasting-furnaces, 16 ft. diam. and 21 ft. high; 5 blast-furnaces (56 by 180 in. at the tuyeres and 14 ft. 10 in. high); 14 matting-reverberatories (hearth 20 ft. by 50 ft. 2 in.) and 8 converter-stands (barrels 12 ft. 5 in. long and 8 ft. in diam.). The fumes from these four departments are drawn off by 4 steel stacks, 200 ft. high, and lined throughout (inner diam., 20 ft.; outer, 23 ft. 6 in.). Since fumes damaged crops in the Deer Lodge valley, it has been decided to deliver them 500 ft. higher up into the air. The fumes from the 4 departments will be conducted through branch-flues into one brick main, 2300 ft. long, built on a hillside, and ending in a stack 300 ft. high and 30 ft. inner diam. The main will be 37 ft. deep; the first 1300 ft. will be 60 ft. wide, and the remaining 1000 ft. 120 ft. wide. Underneath will be a tunnel to receive the cars, into which will be discharged the flue-dust. It is expected that this addition will be finished this summer.

*Boston and Montana Consolidated Copper- and Silver-Mining Co.'s  
Great Falls Plant.*

Ground was broken in the spring of 1890. The plant first consisted of 24 Brückner roasters (cylinders 8 by 16 ft.), 8 tilting reverberatory matting-furnaces (hearth 13 by 16 ft.) and two 5-ton upright converters. The matting-furnaces had regenerative chambers and were fired with producer-gas, resembling Campbell's open-hearth steel-furnaces. The plant has been greatly enlarged since 1890, under the management of F. Klepetko, and the furnaces have been changed to meet constant demand for larger capacity. The present plant comprises 24 Brückner cylinders (8 by 16 ft.), all idle except 2 or 3 used for drying concentrator-slimes; 18 6-hearth MacDougall-Evans-Klepetko roasting-furnaces (16 ft. diam. and 21 ft. high); 5 gas-heated stationary matting-reverberatories with regenerative chambers, 4 of which have hearths 42 ft. 6 in. by 15 ft. 9 in., and one a hearth 29 by 12 ft.; 5 blast-furnaces (18 ft. high to charging-floor; 4 are 56 by 180 in., and one 44 by 180 in. at tuyeres); twelve 5-ton upright converters (7 ft. diam., 14 ft. 7 in. high); one anode-casting reverberatory (hearth 14 ft. 8 in. by 22 ft. 6 in.) with Walker casting-machine; and an electrolytic copper-refinery with 312 tanks and 4 copper-refining furnaces of different sizes.

TABLE I.—Analyses of Butte Ores.

Plant.	A.										B.		C.		D.				
	First-Class Ores.	First-Class Ores.	First-Class Ores.	Second-Class Ores.	Coarse Concentrates.	Coarse Concentrates.	Fine Concentrates.	Fine Concentrates.	Slimes.	Slimes.	Average.	First-Class Ores.	Second-Class Ores.	Concentrates.	First-Class Ores.	Concentrate.			
Cu .....	15.15	21.35	13.80	8.60	8.05	5.9	15.85	15.60	13.45	10.80	9.50	7.20	4.80	7.00	12.7	5.2	8 7	9-10	10
Fe.....	19.50	10.75	.....	18.25	12.10	.....	21.95	22.00	25.90	22.60	22.00	8.40	5.10	21.00	18.4	16.0	27.0	16-20	30
Zn .....	trace.	0.50	.....	0.55	.....	.....	0.50	.....	trace.	.....	trace.	.....	trace.	12.00	.....	.....	.....	.....	.....
S .....	28.93	20.30	.....	23.50	15.37	.....	29.10	29.70	34.70	29.50	28.46	11.70	7.40	32.00	21.8	19.8	31.7	20-25	34-39
Mn .....	trace.	.....	.....	.....	trace.	.....	trace.	.....	trace.	.....	trace.	.....	trace.	.....	.....	.....	.....	.....	.....
As.....	1.70	2.10	.....	0.70	1.26	.....	0.95	.....	0.40	.....	0.90	.....	0.10	.....	.....	.....	.....	.....	.....
Sb.....			.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Pb .....	trace.	.....	.....	trace.	.....	.....	trace.	.....	trace.	.....	trace.	.....	trace.	.....	.....	.....	.....	.....	.....
Al <sub>2</sub> O <sub>3</sub> .....	7.00	3.15	.....	6.15	10.60	.....	3.00	.....	5.70	.....	7.60	.....	20.70	.....	4+	.....	.....	.....	.....
CaO .....	0.28	0.50	.....	0.30	0.54	.....	0.45	.....	0.60	.....	0.00	.....	trace.	.....	.....	.....	.....	.....	.....
MgO .....	trace.	.....	.....	trace.	.....	.....	trace.	.....	trace.	.....	trace.	.....	trace.	.....	.....	.....	.....	.....	.....
Insol.....	30.35	42.25	.....	47.20	56.50	.....	30.30	30.40	21.60	34.00	35.90	66.66	76.50	23.00	40.0	56.0	25.4	45-50	22-24
SiO <sub>2</sub> .....	25.50	36.80	.....	39.30	49.70	.....	26.50	.....	18.15	.....	29.50	.....	53.45	.....	.....	.....	.....	.....	.....
H <sub>2</sub> O.....	.....	.....	.....	.....	.....	.....	.....	1.60	4.7	5.80	.....	29.80	.....	.....	.....	.....	.....	.....	.....
Ag. oz. p. t. ....	8.70	5.50	.....	3.00	3.90	.....	5.00	6.10	.....	4.70	4.00	3.40	2.80	14.00	7.6	3.0	5.6	.....	.....
Au. oz. p. t. ....	0.014	.....	.....	0.016	0.010	.....	0.02	.....	.....	.....	.....	.....	.....	.....	0.082	none.	0.0025	.....	.....

NOTE.—Analyses under A to F, inclusive, represent 1902; those under G, H, I and J are Mr. Goodale's figures for 1896.

TABLE I.—Analyses of Butte Ores.—Continued.

Plant.	E.			F.		G.		H.		I.			J.	
	First-Class Ores.	Coarse Concentrates.	Fine Concentrates.	Silver-Ores.	Silver-Ores.	Smelting.	Concen- trating.	Smelting.	Concen- trating.	Smelting I.	Smelting II.	Concen- trating.	Smelting.	Concen- trating.
Cu.....	8.6	6.0	9.0	0.40	0.30	20.26	6.50	9.80	4.60	29.95	26.25	8.10	0.55	0.10
Fe.....	13.2	18.6	1.42	7.90	5.90	14.42	9.55	26.00	12.10	18.65	16.40	10.10	5.10	4.90
Zn.....	.....	.....	.....	22.50	11.90	5.30	8.66	.....	.....	7.25	2.94	.....	16.29	4.60
S.....	18.0	23.0	34.0	21.00	9.70	26.34	15.87	33.60	15.70	36.15	29.18	14.88	13.19	6.43
Mn.....	.....	.....	.....	1.70	7.43	0.45	0.09	.....	.....	0.26	.....	.....	6.73	5.56
As.....	.....	.....	.....	0.10	0.20	1.66	0.94	.....	.....	0.44	4.00	.....	trace.	.....
Sb.....	.....	.....	.....	8.20	3.50	trace.	0.20	.....	.....	.....	.....	.....	0.05	.....
Pb.....	.....	.....	.....	2.90	7.70	trace.	0.25	.....	.....	.....	.....	.....	8.55	2.44
Al <sub>2</sub> O <sub>3</sub> .....	.....	.....	.....	2.10	4.00	2.35	.....	.....	.....	.....	.....	.....	.....	.....
CaO.....	.....	.....	.....	0.80	1.80	.....	.....	.....	.....	.....	.....	.....	.....	.....
MgO.....	.....	.....	.....	30.40	51.00	28.72	54.80	28.70	65.20	7.26	21.50	63.00	43.22	66.50
Insol.....	54.0	45.0	23.0	26.20	41.70	.....	.....	.....	.....	.....	.....	.....	.....	.....
SiO <sub>2</sub> .....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
H <sub>2</sub> O.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Ag. oz. p. t....	6.5	.....	.....	24.20	67.50	30.	10.	11.2	3.3	24.	20.	6.5	43.	20.
Au. oz. p. t....	n. d.	.....	.....	0.60	0.13	\$0.44	\$0.44	n. d.	n. d.	n. d.	n. d.	n. d.	\$0.80	\$0.40

NOTE.—Analyses under A to F, inclusive, represent 1902; G, H, I and J are Mr. Goodale's figures for 1896.

The early history of this company may be summarized by the dates of the beginnings of operations: Concentrator, March, 1891; Brückner cylinders, April, 1892; reverberatory matting-furnaces, April, 1892; converters, August, 1892; blast-furnaces, April, 1893; refining-furnaces, January, 1893; electrolytic refinery, February, 1893.

In concluding these introductory memoranda, it should be



added that nearly all the smelting-works have concentrating-works, in which second-class ore is enriched.

### III. THE ORES.

The ores of the Butte district are classified by Goodale\* as copper-silver ores, silver-copper ores and silver-ores. The copper-bearing minerals of the copper-silver ores are chalcocite, bornite, enargite and chalcopyrite; the gangue is quartz and feldspar. The copper- and silver-bearing minerals of the second-class are wurtzite, sphalerite, bornite, chalcopyrite, pyrite, tetrahedrite, enargite, tennantite and galenite; the gangue is quartz and feldspar, with some barite and fluorite. Of the silver-ores, which are of secondary importance, the silver-bearing minerals are pyrite, sphalerite, galenite, tetrahedrite and argentite; the gangue is quartz, rhodochrosite and rhodonite. With regard to metallurgical treatment, the copper-bearing ores are divided into first- and second-class. The line of division varies with the different mines. In general, first-class ore contains from 10 to 15 per cent. copper and is smelted direct, usually in blast-furnaces, without having been roasted. Second-class ore, comprising about 90 per cent. of the ore mined, and containing from 3 to 6 per cent. copper, is concentrated; fine concentrates are always roasted and, as a rule, smelted in the reverberatory matting-furnaces; coarse concentrates, above 5 mm. in size, are often added to the blast-furnace charges without having been roasted, the proportion amounting to from 30 to 35 per cent. of the charge.

The partial analyses of 1902, shown in Table I., on pages 268 and 269, taken from smelting-records, give the general character of the ore now treated. The analyses of 1896, given by Goodale† and here quoted for comparison, show much higher values.

### IV. METALLURGICAL TREATMENT—THE ROASTING OF ORES.

The metallurgical processes in the Butte district show much uniformity with regard to the general principles on which they are based. They comprise roasting, smelting, converting, electrolytic-refining and casting. There is considerable diversity in the details of practice.

\* *Trans.*, xxvi., 599.

† *Op. cit.*, p. 639.

First-class ore, forming about 10 per cent. of the product of the mines, is, as a rule, smelted raw, and furnishes the desirable coarse ore of the blast-furnace charge. Concentrates, coarse limestone and converter-slag are the basic fluxes added to slag the siliceous gangue. In one instance, some first-class ore is roasted in lump-form in stalls. This makes it less refractory than raw ore, and, furnishing available iron, permits the running of slags richer in iron, and hence more fusible than would be otherwise possible.

Heap-roasting, which was practiced in the early days, has been entirely abandoned.

In crushing, the copper-silver ores readily break into small grains, making often more fines than can be carried by a stall or even by a blast-furnace. This necessitates separating part of the fines by screening, the screenings going into the fine-ore roasting-furnaces.

The form and general arrangement of the open-stalls now in operation are the same as given by Peters.\* The management, however, differs slightly from the description given by him. In making the bed of decayed wood, first used by A. J. Schumacher at the Parrot works, no attention is paid to an open cross-channel to start the fire.† Peters prefers to close the front of the stall with large pieces of lump-ore; but at Butte it is closed down to 2 ft. from the bottom with a plate of sheet-iron, braced by iron cross-bars to prevent bulging, caused by the swelling of the ores while roasting. The plate is supported by blocks of wood, between which are placed small sticks and kindling. During the roast, when the contents of the stall has risen to the highest point and begins again to shrink on account of the slight clinkering which always takes place, the front plate gradually glides down to the floor. A stall holding 30 tons of first-class ore, with about 10 per cent. of copper and from 30 to 35 per cent. sulphur, burns 30 days, the sulphur being reduced to 3.5 or 4 per cent.

Crushing and sampling the ore, running it to the stalls, and filling and emptying the stalls are done by contract.

The fine-ore furnaces are represented by four types, the Allen-

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\* *Modern Copper-Smelting*, 1895, p. 148.

† See Peters, *op. cit.*, Fig. 21, p. 150.

O'Hara, the Wethey, the Pearce and the MacDougall; long-hearth reverberatory-furnaces, used in early times, having been given up entirely.

*The Allen-O'Hara Furnace.\**

This is one of the permanent improvements on the original O'Hara furnace.† The two superimposed hearths are each 9 by 90 ft.; the 6 carriages, with their plows, make the circuit in  $3\frac{1}{4}$  minutes. Thus the ore, lying about 3 in. deep, is stirred every 37 seconds, and remains about 6 h. in the furnace. A test showed that 3.64 H.P. was required to run the furnace. The ore charged is a mixture of concentrates and fines from first-class ore, usually in the proportion of  $4\frac{1}{2} : 1$ , containing about 35 per cent. of sulphur.

From 47 to 50 tons of charge is roasted in 24 h., or 60 lb. per sq. ft. of hearth-area, the sulphur being reduced to 8 per cent. One hundred pounds of moist ore give 72.7 lb. calcines and 1 lb. flue-dust. A furnace is attended in an 8-hour shift by  $\frac{1}{2}$  car-man, who brings the ore;  $\frac{1}{2}$  fireman; 1 wheeler, who removes the calcines to the smelting-department; and  $\frac{1}{8}$  repair-man, during the day-shift only. A partial analysis of an average sample of roasted ore gave: Cu, 10.8;  $\text{SiO}_2$ , 31.6; Fe, 38.0; S, 7.6; Ag, 7.3 oz. per ton. Flue-dust gave: Cu, 10.8;  $\text{SiO}_2$ , 34.4; Fe, 36.5; S, 8.2; Ag, 7.2 oz. per ton.

*The Wethey Furnace.‡*

The furnaces in operation each have 4 superimposed hearths. The furnaces erected in 1894 had 2 sets of hearths, each 50 by 5 ft., with the rabble-carriages running between them, the arms projecting from the inner sides into the hearth. The furnaces erected since 1894 have been made 65 by 10 ft. in size, with the rabble-arms protruding on both sides and carried on carriages on the outside of the furnace-hearths. The tripping-doors, which close the continuous hearth-slot, are not being repaired as they wear out, the furnace seeming to

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\* Peters, *Modern Copper-Smelting*, 1895, pp. 200–205. Hofman, *Metallurgy of Lead*, 1899, pp. 186–190.

† Küstel, *Roasting Gold- and Silver-Ores*, San Francisco, 1880, p. 84. Fraser and Chalmers' *Catalogue*, No. 4, 1891, p. 135.

‡ Hofman, *Metallurgy of Lead*, 1899, pp. 195–198.

do satisfactory work without them. The position of the fire-box, which was built at first to heat the second floor, has been changed so that the flame may enter the top hearth and dry and kindle the moist concentrates and slimes as they enter the furnace. While the smaller-size furnace, with a hearth-area of 2,000 sq. ft., puts through at the present time 60 tons of raw material with 35 per cent. sulphur, or 60 lb. per sq. ft. of hearth-area, reducing the sulphur to 8 per cent., and consuming 5.5 per cent. of slack-coal, the larger furnace, with an area of 2,600 sq. ft., roasts in 24 hours 90 tons of ore, with 35 per cent. of sulphur, or 70 lb. per sq. ft. of hearth-area, reducing the sulphur to from 5 to 6 per cent., and consuming somewhat less than 4 per cent. of slack-coal. One fireman attends each furnace.

The improvement in the work of the larger furnaces is due to the heat from the fire-box having to travel a longer distance before it escapes to the flue, and to the ore traveling in a direction opposite to that of the flame, whereby the heat is more fully utilized than with the shorter furnace.

#### *The Pearce Turret-Furnaces.\**

Three kinds of turret-furnaces are used in roasting ores, the single- and double-deck furnaces of R. Pearce and the 6-deck furnace of R. F. Pearce. A single-deck furnace, 36 ft. in diam., with a 6-ft. hearth (or a hearth-area of 505 sq. ft.) and two stirrer-arms, roasts, in 24 h., 14 tons of concentrates (Cu, 7.0 per cent.; Fe, 24.0 per cent.; Zn, 12.0 per cent.; S, 32.0 per cent.;  $\text{SiO}_2$ , 23.0 per cent.; Ag, 14 oz. per ton) reducing the sulphur to from 7 to 8 per cent., and burning 400 lb. of coal per ton of ore.

The percentage of flue-dust is very low.

The 36-ft. furnace does better work than the first one built, which has the same width of hearth (6 ft.), but is only 30 ft. in diameter. If, nevertheless, the tonnage appears to be comparatively low, and the percentage of sulphur left in the roasted ore somewhat high, this must be attributed to the large amount of zinc present.

In the double-deck furnaces in which the ore, fed on the

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\* Peters, *Modern Copper-Smelting*, 1895, pp. 205-214; Hofman, *Metallurgy of Lead*, 1899, pp. 175-182.

upper hearth, drops through a slot in the roof onto the lower hearth, the floor of the upper hearth and the roof of the lower hearth are separated, it will be remembered, by an air-space 2 ft. high. A furnace with a 6-ft. hearth (area, 1,010 sq. ft.) treats in 24 h. 30 tons of concentrates (59 lb. per sq. ft. of hearth-area) containing Cu, 12 per cent.; Zn, 8 per cent.; S, 35 per cent.; Fe, 25 per cent.; As, 3 per cent.;  $\text{SiO}_2$ , 16 per cent.; Ag, 20 oz. per ton, with a reduction of the sulphur to from 6 to 7 per cent.; 400 lb. of coal being burned per ton of ore.

A furnace with a 7-ft. hearth (area, 121,829 sq. ft.) treats 42 tons (69 lb. per sq. ft. of hearth-area) with a similar elimination of the sulphur and a fuel-consumption of 182 lb. per ton of ore treated.

Comparing the quantities (59 and 69 lb.) roasted per sq. ft. of hearth-area in the two furnaces, an improvement of 15 per cent. in favor of the 7-ft. hearth is seen. The degree of desulphurization is about the same in both furnaces; the consumption of fuel, however, has been reduced from 400 to 182 lb., or 55 per cent. The labor required to fire a single-deck furnace is the same as with the double-deck, and that for supplying ore and fuel and removing roasted ore and ashes from 1 double-deck furnace is, of course, smaller than from 2 furnaces with single hearths. Thus the double-deck furnace is a decided improvement over the single-hearth as long, be it understood, as ores are treated which do not frit or form accretions on the hearth.

The six-deck furnace\* resembles the double-deck furnace in the stirring mechanism, and in the passage of ore and gases; it has one fire-place (3 ft. by 3 ft. 2 in.) on the top-hearth, and one 4 ft. 3 in. by 3 ft. 2 in. on the bottom-hearth.

It differs from the double-deck furnace in the number of hearths, 6 vs. 2; in the fact that there are no air-spaces between the superimposed hearths, the roof of one forming the floor of the next one above; and that the portions of the inner wall above the horizontal slots of hearths 1 to 5 have cantilever supports with trussed chords projecting from the central column. This makes rather a complicated structure, requiring a large number of iron parts of various sizes, some of considerable

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\* *U. S. Patent*, No. 671,071, April 2, 1901.

weight. The iron parts of the furnace weigh 165 tons. The furnace is 32 ft. in diam. and 20 ft. 9 in. high; the hearths, 7 ft. wide, are 2 ft. 9½ in. from roof to roof; roofs, 4½ in. thick; the sand-filling forming the working-hearth is 5 in. thick at the crown of the roof and 12 in. at the sides; the distance from the working-hearth to the inner slot (protected by 1-in. castings) is 6 in., and to the springings of the arch 16 in.; the rise of the arch is 8 in. A hearth has 4 stirring-arms making a revolution in 65 seconds. The horizontal flues in the outer walls are omitted, through which air passes in the older furnaces, in order to be warmed and admitted to the surface of the roasting ore. Around the furnaces are platforms to facilitate access to the hearths. The moist concentrates, dumped into the shoe of a bucket-elevator, are raised to the top of the furnaces and discharged into a hopper, partly closed at the bottom, by a corrugated roller-feed; this delivers into a slot, 8 in. wide by 7 ft. long, in the roof of the top-hearth. The furnace roasts 56 tons ore in 20 h., or 38 lb. per sq. ft. of hearth-area; it makes 4 per cent. of flue-dust on the weight of the ore charged, and requires 28.5 lb. slack-coal per ton of ore. The quality of the work is similar to that of the double-deck furnace; the amount of ore-roasted per sq. ft. of hearth is a great deal smaller, but the saving of fuel is much larger. The low fuel-consumption brings out in a striking way the benefits that are derived by diminishing the loss of heat by radiation.

### *The MacDougall Furnace.*

This is a vertical cylindrical furnace, with arched horizontal hearths having discharge-openings alternately near the center and periphery, and a central revolving shaft with horizontal radial stirring-arms, which are provided with teeth placed at an angle to the arms. The teeth of the arms on the even-numbered hearths are set in a direction opposite to that of the odd-numbered: thus, on the odd-numbered hearths, they push the ore toward the center; and on the even-numbered, toward the side. The ore, fed mechanically into the furnace, is delivered continuously onto the outer edge of the first hearth, where the teeth of the stirring-arm turn it over and work it toward the center, where it drops through an annular opening onto the second hearth; the arms of this turn the ore over and over

and convey it slowly to the openings near the periphery, through which it drops onto the third hearth, and so on, until it arrives on the closed bottom, where it falls through drop-holes into a hopper.

The ore, spread evenly over the hearth, travels in a zig-zag path through the furnace, passing from periphery to center and *vice versâ*. It is warmed at the top of the furnace: then it ignites and burns freely; and, lastly, most of the sulphates formed are decomposed lower down. The air necessary for oxidation is admitted through doors in the side, most of it entering on the bottom hearth. The gases, and with them the dust, pass upward through the ore-discharge openings and are drawn off through pipes in the roof. In starting, the furnace has to be heated up to the kindling temperature of the ore, which, if rich enough in sulphur, will burn of its own accord without the use of any carbonaceous fuel. If too low in sulphur, the heat necessary for successful roasting has to be supplied by one or more external fire-places.

Three kinds of MacDougall furnaces interest us here:—The Original MacDougall, the Herreshoff and the Evans-Klepetko.

*The Original MacDougall Furnace.\**—This furnace, patented by MacDougall Brothers in the beginning of the seventies, was in operation in Liverpool, England, burning pyrites to furnish sulphurous gases for the manufacture of sulphuric acid. It had to be abandoned on account of the large amount of flue-dust formed and the many mechanical difficulties encountered. The furnace, 6 ft. in diam. and 12 ft. high, was built of 7 cast-iron cylinders bolted together and had 6 closed chambers. The vertical solid shaft, 6 in. in diam., had cast-iron stirring-arms. A 2-H.P. engine was sufficient to drive the mechanism. The furnace roasted in 24 h.  $3\frac{1}{2}$  tons pyrite, or 0.02 tons per sq. ft. of hearth-area, reducing the sulphur to 2 per cent.

*The Herreshoff Furnace.*—Most of the Herreshoff furnaces in operation at Butte have a  $\frac{1}{4}$ -in. steel casing; they are 10 ft. 10 in. in diam. and 11 ft. 6 in. in height, and are lined with a full course of red brick. A few furnaces, built of two full courses of brick and hooped with iron bands only, are doing better

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\* Lunge, *Sulphuric Acid and Alkali*, London, 1891, vol. i., pp. 260-264. Also, *The Mineral Industry*, 1898, vol. vi., p. 236.

work in cold weather than the furnaces that are sheeted and lined with a single course, thus showing that with ores running low in sulphur, when heat has to be husbanded, an 18-inch side-wall has its advantages. A furnace has 5 arched brick hearths,  $4\frac{1}{2}$  in. thick and 12 and 13 in. apart; the pitch of the arch is  $4\frac{1}{2}$  in.; a current of air passes through the hollow central revolving-shaft, which is 14 in. in diam. The hollow horizontal cast-iron stirring-arms, 2 to a hearth, have 7 and 8 teeth respectively; they are easily exchanged when worn out.

The top hearth acts as a dryer; the second starts the roasting; on the third the ore roasts freely (innumerable sparks are seen); on the fourth there are no sparks; and on the fifth, the ore looks dark. A furnace, with its shaft making 50 revolutions per hour, roasts in 24 h. from 5 to 6 tons of wet concentrates with about 35 per cent. of sulphur, or 0.015 tons per sq. ft. of hearth-area, reducing the sulphur to 6 per cent. Seventy furnaces make from 4 to 5 tons, or 6.4 per cent., of flue-dust.

In Eastern sulphuric-acid plants, this furnace\* treats in 24 h. from 3 to 4 tons of pyrites, with 44 per cent. of sulphur, or 0.0098 tons per sq. ft. of hearth-area, reducing the sulphur to from  $2\frac{1}{2}$  to  $3\frac{1}{2}$  per cent.. The shaft makes 30 revolutions per hour.

*The Evans-Klepetko Furnace.*†—Shortly after the Herreshoff furnace had proved a success, a larger furnace,‡ 16 ft. in diam., was constructed on the same general lines; it had 8 hearths, an air-cooled shaft and 8 solid cast-iron arms. The heat generated was sufficient to cause the arms to bend under their own weight. Air-cooling having proved unsatisfactory, water-cooling was substituted, and the number of hearths reduced from 8 to 6. The leading differences, then, between the Evans-Klepetko and the Herreshoff furnaces are the size and the water-cooling of the central shaft and the stirring-arms. The furnace is 18 ft.  $3\frac{1}{2}$  in. high and 15 ft. 10 in. in diam.; it is sheathed with  $\frac{3}{8}$ -in. boiler-iron and lined with a full course of red brick; it has 6 arched hearths with a 9-in. spring and 3 ft. apart; each

\* *The Mineral Industry*, vol. vi., pp. 235-238.

† U. S. Patent, No. 700,339, May 20, 1902; Drawing, see *Canadian Mining Review*, April 30, 1903.

‡ C. H. Repath, "The Mechanical Engineer, a Factor in Modern Mining, Milling and Smelting," Mining Congress, Butte, September 3, 1902, through *Mining and Scientific Press*, September 13, 1902.



hearth has 2 stirring-arms making 60 circuits per hour. The furnace stands on columns 12 ft. high, to allow the roasted ore to be collected in hoppers and discharged into cars. Six (or 8) furnaces form a battery; they are placed 18 ft. apart from center to center in one direction and 21 ft. 3 in. in the other. Each furnace has 2 gas-flues, 2 ft. in diam. and 12 ft. apart, passing out of the roof; the flues from 3 furnaces lead into 1 main 6 ft. (or 7 ft.) in diam., having openings along the top and bottom for removing the flue-dust; the bottom openings have vertical pipes which are connected directly with the calcine-hoppers. The central shaft of a furnace is driven from below. Three (or 4) furnaces receive their motive power from one main shaft, with which they are connected by friction clutches. A clutch can be thrown in or out from each of the 3 platforms surrounding a furnace. The cooling-water is forced down to near the bottom of the revolving hollow shaft (9 in. inner diam.) through a 3-in. pipe and out to the ends of the horizontal stirring-arms through 1-in. horizontal pipes. In its upward passage between shaft and pipe it takes up the return-water from the stirring-arms and discharges at the top through 2 spouts into a stationary launder. Shaft and arms are made up of flanged sections to permit easy exchange. Running the overflow-water at 80° C., 20 gallons of cooling-water per minute are required by a furnace. Forty-eight furnaces are supplied with the necessary draft by a steel stack 200 ft. high and lined throughout (inner diameter, 20 ft.; outer, 23 ft. 6 in.). A 30-H.P. motor is provided to run the machinery. Tests have since shown that 10 H.P. is required for 6 furnaces. When ore, too low in sulphur to be self-burning,—*i. e.*, about 28 per cent.,—is to be roasted, one auxiliary fire-place for 2 furnaces, placed on the level of the bottom floor and delivering the flame onto the fourth floor, furnishes the additional heat required.

The moist concentrates arriving from the ore-dressing plant on an overhead track are dumped into the 2 feed-hoppers of a furnace. These are 10 ft. apart and are 4 ft. 6 in. in diam. for 9 ft. 6 in. and then conical for 7 ft. 6 in. At the bottom they are made 3 ft. 6 in. by 1 ft. 2 in., to fit into the openings in the roofs of the furnaces. The top of the hopper has a grating to break up any lumps and to keep out large pieces that may have accidentally got into the ore. The hopper holds 2 cars of

concentrates, or 33 tons. This weight, pressing downward, prevents any hanging of the charge and thus solves the difficulty of feeding moist fine concentrates. The contents of the hopper is continuously discharged into the furnace by means of a feeding device of the form of an I, which passes to and fro under the mouth with a throw of about 10 in. The rods of the two feeders are joined by a cross-piece which is pivoted in the center and connected by a rod to one end of a link, while the other end is attached to the pitman of a crank and spur-wheel; the spur-wheel is geared to the central shaft of the furnace. The ore is spread on the top hearth to the thickness of 3 in. by the stirring-arms. The tops of these arms are protected by cast-iron caps against rapid wear at the places where they strike the ore-column coming down through the feed-openings. The outer half of a hearth is 9 in. thick, the inner 6 in. The reasons for the difference are, saving of brick, reducing of weight at center, preventing contact with stirring-teeth when the hearth rises upon heating, and furnishing an offset to hold the loose working bottom (crushed limestone, sometime flue-dust and even coke-dust) which is about 4 in. thick at the center and 9 in. at the periphery. The two arms of a hearth have 7 and 8 cast-iron teeth; these are 8 in. long by 6 in. wide by  $\frac{5}{8}$  in. thick; the lower 3 in. of the teeth, which come in contact with the ore, are chilled. There is no advantage in making the teeth thicker than  $\frac{5}{8}$  in., as they wear off obliquely to the working surface and, consequently, would simply offer a larger wearing surface but not lengthen the life. At the same rate that the teeth are worn off, the ore builds on the loose working-hearth. When they are to be renewed, they are removed and a plow is slipped over the stirrer-arm and moved a little toward the center after every circuit of the arm. Thus a crust is easily and surely broken up. The crusts on the second hearth are harder to break than those on the other hearths. The teeth on the top hearth last from 25 to 34 days; those on the sixth, from 6 to 8 months. The shorter life of the former is attributed to mechanical wear caused by decrepitation of the ore. It cannot be due to chemical action, as the top hearth serves mainly for drying, unless the moisture should cause some sulphuric acid to form; but the temperature of the escaping gases is 315° C. and the draught is sufficiently strong to show a depression 0.3 in. of water.

Roasting begins on the second or third hearth, varying with the strength of the draught and the number of circuits that the stirring-arms make. With a strong draught the heat creeps up; at one circuit in 55 seconds the roast begins on the second hearth; at one in 75 seconds on the third. As the ore passes from the first hearth through the annular opening at the center of the roof of the second hearth (protected by a cast-iron ring), it strikes a distributor which prevents any of it from passing down the shaft. During normal work three doors on the bottom hearth are left open for the admission of air. If the furnace becomes too hot, the doors on the third floor are opened more or less and thus the draught is checked. While the ore drops from hearth to hearth, the upward gas-current carries with it dust, and this striking the roof adheres to it in part and accumulates. In order to protect the arms from being worn out at these places, they are protected by cast-iron caps. When the dust has grown to a thickness of 4 in. it is removed with chisel-pointed bars. Experiments with cast-iron plates to protect the roof at weak spots have proved successful, as the dust is easily pried off. The chunks of fritted dust go to the blast-furnace as welcome lump-ore; they contain only 6 per cent. of silica with ore of 30 per cent. silica, thus showing that light particles of ferric oxide are more easily carried away by the gas-currents than heavy gangue. The dust collected in the settling-chambers runs higher in sulphur than the roasted ore.

In starting a furnace, crushed limestone is fed to form the working-bottom. Then a small fire of dry, soft, *i.e.*, long-flame, wood is started from the three side doors of the third and fifth hearths. A new furnace is brought to dark-red in from 3 to 4 days; an old one requires only two days. Now, concentrates are fed. After charging for from 5 to 6 hours, it sometimes happens that the furnace cools down too much. This makes it necessary to start on the third and fifth floors a new fire for from  $1\frac{1}{2}$  to 2 h.; occasionally, feeding of the ore is stopped and half a ton of coal is charged. When this coal has come to the third hearth, the machinery is stopped, and it is allowed to burn out. The kindling of the coal may have to be assisted by placing dry wood on the hearth. Under normal conditions a furnace does its best work when the flue shows a depression in water of 0.3 in.; if it is less, the furnace gets cool. The

normal ore-mixture passes through the furnace in about  $1\frac{1}{2}$  h. If the furnace gets too hot, more slimes are fed; if too cold, coarser concentrates are charged and the rate of feeding is reduced. As indicated above, the temperature is further regulated by the admission of air; closing the bottom doors drives up the heat, opening them draws it down; opening doors higher up checks the draught. The rate of feed when once settled upon is usually not altered, and the number of circuits the arms make per hour remains, of course, constant.

The dust, which collects in the flue connecting 3 (or 4) furnaces and forms from 4 to 5 per cent. of the ore, is raked out every day. Sometimes part of it is blown into the main flue of a battery, with a hose under a pressure of about 90 lb. per sq. in. The loss of weight in ore including flue-dust is about 20 per cent.

A furnace treats in 24 h., under normal conditions, 40 tons of sulphide-ore, with 35 per cent. sulphur and 10 per cent. copper, or 0.042 tons per sq. ft. of hearth-area, reducing the sulphur to 7 per cent.; roasted-ore, with 14 per cent. copper treated in the same manner, retains about 10 per cent. of sulphur. The amount treated can, of course, be varied with the speed of travel of the stirring-arms. Thus, with one circuit in 75 seconds it is as low as 30 tons in 24 h.; with one in 50 seconds it is increased to 50 to 55 tons.

A section of 6 furnaces, or even 8, is tended in an 8-hour shift by  $\frac{1}{3}$  foreman, 1 furnace-man, 1 helper,  $\frac{1}{8}$  oiler,  $\frac{1}{8}$  repairman and 1 trimmer.

The following average partial analyses of roasted-ore represent two determinations from the average day-and-night samples, taken during an experimental run of 15 days:

SiO<sub>2</sub>, 26.9 per cent.

Cu, 18.3 per cent., of which 9.9 was present as CuO.

Fe, 30.0 per cent., of which 17.9 was present as FeO.

S, 9.2 per cent., of which 0.81 was present as SO<sub>3</sub>.

The matte formed in crucible fusions contained 65 per cent. of copper. In comparing the roasted-ore of the MacDougall furnace with that of the Brückner furnace\* of the same plant, it is found that it is reddish-brown as against a dark-brown (more Fe<sub>2</sub>O<sub>3</sub> is formed), more porous (1 cu. ft. weighs 87 lb. as against 100 lb.) and finer (the rolling of the ore in the Brückner cyl-

\* Size: 8 by 16 ft., 1 revolution in 7 min.; 15-ton charge in 24 h.; drying, 2 h.; burning, 4 h.; charging, roasting and discharging, 18 h.

inder causes particles to form pellets). In regard to flue-dust, the MacDougall furnace makes much less than the Brückner cylinder, although the amount is still great: they compare as 1:3 to 4.

*General Results.*—Some of the leading facts of the roasting-furnaces operated in Montana are shown in Table II.:

TABLE II.—*Details of Montana Roasting-Furnaces.*

	Tons Roasted in 24 hrs.	Horse-Power Required.	Sulphur in Concen- trates.	Sulphur in Cal- cines.	Area of Hearths.	Concentrates per Sq. Ft. Hearth.	Coal per Ton Con- centrates.	Cost per Ton.
			Per Cent.	Per Cent.	Sq. Ft.	Lb.	Lb.	
Hand-reverberatory furnace, 69½ ft. x 16 ft. hearth.....	13	.....	35	7 to 8	1112	12	307	\$2.00
Allen-O'Hara fur- nace, two hearths, 94 ft. x 9 ft.....	51	3.64	35	8	1692	77	145	0.78
Brückner cylinder, 8 ft. x 16 ft.....	18-20	1½	37	9.5	.....	.....	540*	1.25
Pearce single-deck furnace.....	14†	1½	32	7 to 8	505	55	400	0.98†
Pearce double-deck furnace, 6-ft. hearths.....	30†	3	35	6 to 7	1010	59	400	0.98†
Pearce Double-Deck furnace, 7-ft. hearths.....	42†	3	35	6 to 7	1218	69	182	0.98†
Keller-Gaylord-Cole furnace, two sets of six hearths....	50	1¾	38	7 to 10	2592	38	67	.....
Wethey furnace, two sets of four hearths, 50 ft. x 5 ft.....	60	4	40	8	2000	60	110	.....
Wethey furnace, two sets of four hearths, 65 ft. x 10 ft.....	90	4	35	5 to 6	2600	70	80	....
Herreshoff furnace, five hearths.....	5-6	.....	35	6	135	80	.....	0.40
MacDougall-Evans- Klepetko furnace, six hearths.....	40	1¾	35	7	952	84	.....	.35
Pearce multiple- hearth furnace, six hearths.....	56†	12	35	6 to 7	2947	38	28.5	0.98†

\* Data obtained from operations of six months at Great Falls. For quality of coal see analysis of Sand Coulee and Belt coal (p. 294).

† Average.

‡ These low figures are due to the character of the ore (Gagnon mine) which carries from 8 to 12 per cent. of zinc.

## V. SMELTING ORES IN THE BLAST-FURNACE.

The leading data of the blast-furnace practice are brought together in Table III., on page 284. The furnaces are all of the same general type; they are oblong and water-jacketed, and have an internal crucible, which discharges the slag-matte mixture continuously over a raised spout, whereby the blast is trapped. The matte is settled in a large forehearth, while the slag overflows, to be granulated and removed by water, or to be collected in waste-slag cars and conveyed electrically to the dump and poured.

The general tendency is to increase the size of the furnaces. The smallest distance between tuyeres is 42 in.; the largest, 56 in.; in one instance it was increased to 72 in. This worked well with coarse ore; but it proved a failure with the usual run of ores, which contain many fines. Thus, a screen analysis of an average sample of blast-furnace ore from one important mine gave:

300 lb. ore over 2 in., . . . . .	13.3 per cent.
450 lb. ore 2 in. to 1 in., . . . . .	20.0 per cent.
500 lb. ore 1 in. to $\frac{3}{8}$ in., . . . . .	22.2 per cent.
1000 lb. ore under $\frac{3}{8}$ in., . . . . .	44.5 per cent.
<hr/> 2250 lb.	<hr/> 100.0 per cent.

showing why such a large distance between tuyeres proved impracticable. The greatest length of furnace at tuyeres is 180 in.; this seems to be the limit with the present mode of construction of jackets. The height of the furnaces increases with the distances between tuyeres from 7 to 18 ft. But the charge in the 18-ft. furnace is kept from 4 to 8 ft. below the throat, so that the actual working-height is reduced to from 10 to 14 ft. All furnaces are run with a top more or less hot. Temperature-measurements of furnace-gases in one instance showed 315° C., and a draught-measurement gave a depression 0.8 in. of water. The water-jackets are all made of low-carbon steel. They are 2-high, reaching to a short distance below the feed-door, the remaining space being of brick. Usually two jackets are on a side and one on each end; the lower tiers of side-jackets only have a bosh, the amount of which is very small, while the upper tiers are vertical, as are generally the end-jackets. The gases are always drawn off above the feed-floor. In one instance the top

TABLE. III.—*Examples of Montana Blast-Furnace Smelting.*

Furnace.	A.	B.	C.	D.	E.	F.
Horizontal section at throat.....	66 in. x 112 in.	54 in. x 96 in.	72 in. x 180 in.	53 in. x 160 in.	72 in. x 180 in.	72 in. x 180 in.
Area at tuyeres.....	42 in. x 112 in.	44 in. x 96 in.	56 in. x 150 in.	42 in. x 160 in.	56 in. x 180 in.	56 in. x 180 in.
Height, tuyeres to throat.....	92.65 ft.	23.33 ft.	70.029 ft.	46.629 ft.	70.29 ft.	70.29 ft.
Height, tuyeres to top of crucible.....	7 ft. 34 in.	9 ft.	18 ft.	10 ft. 6 in.	18 ft.	18 ft.
Water-jackets height.....	16 in.	9 in.	11 in.	9 in.	11 in.	11 in.
Bosh, inches in feet.....	9 ft.	7 in. in 5 ft.	14 ft. 10 in.	10 ft. 6 in.	11 ft. 10 in.	11 ft. 10 in.
Crucible, depth.....	8 in.	16 in.	8 in. in 7 ft.	1 in. in 10 in. 2	8 in. in 7 ft.	8 in. in 7 ft.
Forehearth, fixed or movable.....	Fixed.	Fixed.	27 in.	9 to 14 in.	28 in.	28 in.
Forehearth, shell dimensions.....	8 ft. x 5½ ft. x 3 ft.	12 ft. dl. 52 in. deep.	11 ft. dl. 56 in. deep.	Fixed.	Fixed.	Fixed.
Tuyeres, number.....	16	12	32	7 ft. 3 in. x 10 ft. 6 in. x 4 ft.	32	32
Tuyeres, diameter.....	3 and 3½ in.	2½ in.	3 in.	18	3½ in.	3½ in.
Tuyere-ratio*.....	3.5	2.01	3.23	4 in.	3.79	3.79
Blast, pressure, oz.....	10†	24	35	30 to 32	28	28
Blast, cu. ft. per ton of ore.....	57,000	68,574	65,000	.....	.....	60,176
Blast, temperature, degrees C.....	150	15	15	.....	15	15
Charge (ore + flux), weight, lbs.....	3800	2400	2100	4000	8700	10,000
Charge, tons per sq.-ft. hearth-area.....	195	165	400	220	100	450
Coke, per cent. ash.....	3.82	5.56	5.71	4.71	5.71	6.43
Coke, per cent. of charge.....	20.0	21.3	9 to 15	20.0	8.0	8.0
Men, number in 8-hr. shift.....	13	12.3	6.1	11.5	9.2	8.5
Mate, per cent. of charge.....	5 + 2	4 + 2	3 x 2½	4½ + 3½	3 +	4 +
Mate, Cu. per cent.....	50	46	49	38	42	5.5
Zn. per cent.....	.....	.....	1.0	.....	.....	42.12
Ag. %.....	69	52	20	.....	.....	.....
Al. %.....	2	10	.....	.....	.....	.....
Sp. gr.....	.....	.....	5.0 to 5.1	.....	.....	.....
Slag, SiO₂.....	47.7	44.3	40.7	48.0	40.0	4.8
Fe(Mn)O.....	18.5	27.0	26.2	17.0	23.0	42.6
Ca(Mg)O.....	28.2	23.0	19.4	20.0	22.6	30.7
Al₂O₃.....	.....	9.0	9.6	6.0	9.8	19.5
Cu.....	0.35	0.32	0.25	0.30	0.30	0.25
Ag.....	0.43	0.30	.....	.....	0.20	.....
Sp. gr.....	.....	.....	3.0 to 3.55	.....	.....	3.3

\* Sq. in. tuyere-area : 1 sq.-ft. hearth-area.

† Estimated at 50 per cent. of the rated capacity of the blowers.

‡ First figure represents the regular crew, second figure auxiliary labor.

§ Lower jackets only.

|| Ore-flux.

of the furnace carries a Giroux blast-heater, made by the Union Iron-Works of San Francisco, which raises the temperature of the blast to  $150^{\circ}$  C., and, at the same time, acts as a dust-catcher, delivering the saved dust back into the blast-furnace. The charge, in most works, is introduced by hand from the sides of the furnace; occasionally it is fed from the ends. In some instances, with large furnaces, putting through 400 tons charge in 24 h., mechanical feeding has taken the place of hand-feeding. The charge is brought by electric or compressed-air traction to the side of the furnace on trucks with tilting-carriages, the doors are raised by compressed air, and the charges slid in alternately from one side and the other. This method seems to be satisfactory. At least, worked out in one large plant, it has been introduced in another with some improvements in the construction of the carriage. While it is generally maintained, and has recently been again emphasized by P. Johnson\* in connection with copper-smelting, that the successful working of a blast-furnace is much governed by the care given to feeding, it must be admitted that, considering the mechanical-feeding practiced at Butte with copper-ores producing siliceous slags high in lime, and at Pueblo, Colo.,† Murray, Utah,‡ East Helena, Mont.,§ with lead-ores, where blow-holes or uneven sinking of charges must be absolutely avoided, that hand-feeding cannot hold its former monopoly, but will have to yield to mechanical devices.

The hearth in some plants is supported by jack-screws; in others it is built up solid from the foundations. It rises some distance above the furnace-floor in order to furnish the height necessary for the forehearth or settler and the disposal of the waste-slag. The hearth is lined to a depth of from 24 to 30 in. with fire-brick, or only with a course or two of fire-brick, and then with a mixture of quartz and clay. The crucibles have a depth varying from 8 to 28 in. Although deep crucibles last rarely longer than 3 months, and shallow crucibles have a longer life, the former are more common. Preliminary experiments with

\* *Engineering and Mining Journal*, August 23, 1902.

† Hofman, *Metallurgy of Lead*, 1901, p. 216; *Trans.*, xxxii., 353.

‡ *Engineering and Mining Journal*, June 28, 1902.

§ *Trans.*, xxxii., 380.



lining deep crucibles with chrome-brick\* have given promising results in preventing break-outs of matte. There appears to be no *à priori* reason for having a continuously discharging-crucible as deep as 28 in.; from 8 to 10 in. would seem to be sufficient to furnish the matte-bath necessary to cover the bottom satisfactorily, but this is disputed by the best of authority. In treating zinkose ores the deep crucible would probably soon fill up with mushy matte and cause infinite trouble, but the ores smelted, fortunately, contain very little zinc.

It is claimed that with a deep crucible a furnace puts through more charges, but this has, so far, not been proved. As far as the strength of the jackets is concerned, the deep crucible is preferable, as with a shallow crucible the tuyeres are placed higher up (viz., 24 and 12 in.; see Table III.), in order to furnish a smelting-zone of some height, and the water-cooled breast has to be built into the jacket; while, with the deep crucible, it is placed underneath the jacket.

The slag-matte mixture is usually discharged from the center of one of the sides of a furnace. The water-cooled tymp is made of cast-iron or, preferably, of copper. Experiments with copper and copper-alloys gave the following data:

Life, Days.	Cu. Per Cent.	Zn. Per Cent.	Al. Per Cent.	P. Per Cent.	Sn. Per Cent.
365	100	.....	.....	Some.	.....
144	90	.....	0.5	.....	10
75	89	3.0	0.25	.....	8

The solid cast-iron spout with water-cooled nose is very common; wrought-iron pipe surrounded by cast-iron has proved unsatisfactory; a water-cooled wrought-iron spout with a cast-copper removable nose lasts about 120 days, and is often preferred to cast-iron.

The tuyere-stock in common use consists of a cast-iron tuyere-box firmly attached to the jacket and connected by a sheet-iron tuyere-pipe, having a gate, with the bustle-pipe.

The large size of the roofless forehearth in which the matte settles out from the overflowing slag is a development of Mon-

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\* Lang, *Engineering and Mining Journal*, January 23, 1897. Packard, *Engineering and Mining Journal*, February 13, 1897. Glenn, *Trans.*, xxxi., 374.

tana practice, caused by the necessity of storing a considerable quantity of liquid matte for the converters. As far as the settling of matte is concerned, oval and oblong hearths are sufficient, as the freedom of copper in waste-slag depends to a large extent upon the length of the path the slag can travel before it overflows. In regard to storage-capacity and strength of vessel, the circular-hearth is preferable, hence it is only with a comparatively small blast-furnace not furnishing matte direct that we find the oval or oblong forehearth. The circular forehearth, 14 ft. in diam. and 56 in. high with a 9-in. brick-lining and a 9-in. brasque backing, holds about 60 tons of matte when new. In order to prevent breaking out, the shell is sprayed with water. After 3 months' use, its capacity has become so decreased by accumulation in the interior that it necessitates renewal. The 9-in. course of brick sometimes has a 3-in. backing of sand which seems to be sufficient. The smaller oblong hearth has, as a rule, cast-iron water-jackets on the sides; it has a 9-in. bottom which crusts up about 6 in., and the sides are lined with a half-course of brick, a 1-in. air-space being left at the water-jacket. All forehearths have, of course, a tapping-slot for matte. It is closed in a few instances by a cast-iron plate, but more commonly by a solid copper-plate with tap-hole. The matte is tapped from the forehearth into a ladle, which discharges its contents either into flat cast-iron moulds, holding about 3,000 lb., or into the converter. The practice of bringing the converter to the forehearth of the blast-furnace has become obsolete. The blast-furnace forehearth, if of large size, is of great assistance to the reverberatory matting-furnaces in a plant which converts matte, as it makes little difference whether the forehearth contains much or little matte; it can be allowed to fill to near the top, or it can be tapped nearly dry. With the reverberatory-furnace this is not the case. The aim there is to allow the level of the matte to fluctuate as little up and down as possible. This means that only a certain quantity of matte can be tapped at stated intervals. If, therefore, the converter calls for matte, and the reverberatory-furnace can furnish only an insufficient amount, there is the blast-furnace forehearth, forming a kind of balance-wheel, to help over the difficulty. The waste-slag is caught in tilting waste-slag pots of from 5 to 6 tons capacity, and hauled to the dump,

or it is granulated. Near the bottom of the launder carrying the slag-granules is often placed a mechanical sampler, driven by the granulating-water, which takes out the whole of the stream of granules at stated intervals and thus furnishes a true sample of all waste-slag made.

### Coke.

The coke used around Butte comes from different parts of the country, as shown in Table IV.; most of it, however, is made from Montana coal:

TABLE IV.—Average Coke Analyses.

Year	Total H <sub>2</sub> O.	H <sub>2</sub> O.	Volatile Matter.	Fixed Carbon.	Ash.	Sp. Gr.
<i>Lehigh, Penn. :</i>						
1897.....	.....	0.3	1.3	85.7	12.6	.....
1898.....	.....	0.5	3.2	82.7	13.4	.....
1899.....	.....	0.4	3.9	81.0	14.6	.....
1900.....	0.6	0.2	4.2	81.8	13.8	.....
1901.....	0.3	0.2	3.6	84.2	12.2	.....
Total .....	0.9	1.6	16.2	415.4	66.6	.....
Average.....	0.4	0.3	3.2	83.1	13.3	.....
<i>Horr, Mont. :</i>						
1899.....	.....	0.5	4.2	75.2	20.3	.....
1900.....	.....	0.3	4.0	75.7	19.9	.....
Total .....	.....	0.8	8.2	150.9	40.2	.....
Average.....	.....	0.4	4.1	75.4	20.1	.....
<i>Crow's Nest, British Columbia :</i>						
1900.....	0.5	0.2	4.1	85.8	9.9	.....
1901.....	0.3	0.2	3.3	85.3	10.7	.....
1902.....	.....	0.2	4.7	84.5	10.5	.....
Total .....	0.8	0.6	12.1	255.6	31.1	.....
Average.....	0.4	0.2	4.0	85.2	10.3	.....
<i>Belt, Mont. :</i>						
1899.....	.....	2.3	4.7	79.3	13.7	.....
1902.....	.....	0.3	3.6	83.0	12.4	1.45
Total .....	.....	2.6	8.3	162.3	26.1	.....
Average.....	.....	1.2	4.2	81.1	13.0	.....
<i>Cardiff, Colo. :</i>						
1902.....	.....	0.4	3.6	85.7	10.3	2.45
<i>West Superior, Wis. :</i>						
1902.....	.....	0.3	3.7	81.4	14.5	.....
<i>Fairmount, W. Va. :</i>						
1902.....	0.1	0.1	4.5	82.4	13.0	.....
<i>Castle Gate, Utah :</i>						
1902.....	.....	0.2	3.5	86.1	10.2	.....

*Analyses of Coke-Ash.*

Source.	Kind.	SiO <sub>2</sub> .	Fe <sub>2</sub> O <sub>3</sub> .	Al <sub>2</sub> O <sub>3</sub> .	CaO.	MgO.	S.
Montana.....	Horr.	57.0	19.0	21.2	3.3	.....	.....
“ .....	Belt.	51.6	10.0	33.6	.....	.....	.....
Colorado.....	Cardiff.	48.9	18.9	23.9	5.8	1.4	0.8

In the management of a blast-furnace the tendency has been toward large tonnages, with the production of 50-per cent. copper-matte, and slag low enough in copper to be a waste-product. Tonnage has had to contend with siliceous ores low in iron, and with limestone as the only available flux, which caused the making of slags of high melting-temperatures. As, further, large amounts of limestone are required to make slags that are at all fusible, the formation-temperatures lie higher than when there is a more intimate contact of acid and base. Tonnage was therefore obtained mainly by increasing the size of the furnace, as shown in Table III., on page 284. The combination of siliceous slags low in iron, with much air, has been the cause of as much as 70 per cent. of the sulphur in the charge being burnt off in its descent through the furnace. But even with the large elimination of sulphur in a comparatively dense charge, it is often not possible to obtain the 50-per cent. matte desirable for the converter-plant, hence blast-furnace matte is sometimes tapped from the forehearth into a ladle and poured into the reverberatory matting-furnace to raise the percentage of copper by mixing with higher-grade matte. On page 290, Table V. gives in percentages some idea of how the charges are made up, and Table VI. shows partial analyses of flue-dust and briquettes, which give an idea of their general character.

The briquetting-machines of Chisholm, Boyd & White (Chicago) and of H. S. Mould (Pittsburg) are both used.

## VI. SMELTING ORES IN THE REVERBERATORY MATTING-FURNACE.

The leading facts of the reverberatory-furnace practice are shown in Table VII., on pages 292 and 293.

The characteristics of Montana reverberatory matting-furnaces are the form of hearth and the greatness of its area. While as late as 1891 the hearth had the usual pear-shaped form of the

TABLE V.—*Percentage of Components of Charge.*

Plant.	First-class Ore.	Coarse Concentrates.	Matte. <sup>*</sup>		Converter Slag <sup>†</sup>	Briquettes. <sup>‡</sup>	Limestone.	Coke.
A.....	22.4	20.5	4.0	17.8	4.0	22.8	8.5	
B.....	52.5	.....	2.3	8.9	.....	25.6	10.7	
C.....	15.5	25.0	22.5		0.2	27.6	9.2	

\* The matte comes in part from the reverberatory matting-furnace, which had to be tapped when the converter-plant was not ready. In part it is purchased matte to be re-melted for the converter.

† The converter-slag always goes to the blast-furnaces, as it furnishes coarse material, aids the smelting, in having been already once smelted, and acts as a basic-flux (see Analyses below). Experience has shown that it not only does little good in the reverberatory matting-furnaces, but that it does harm in that it increases the amount of slag to be skimmed, and reverberatory-furnace slags run higher in copper than blast-furnace slags.

‡ The briquettes are a mixture of flue-dust with 3 or 4 per cent. slacked lime; sometimes slimes from the ore-dressing works are added to replace the lime.

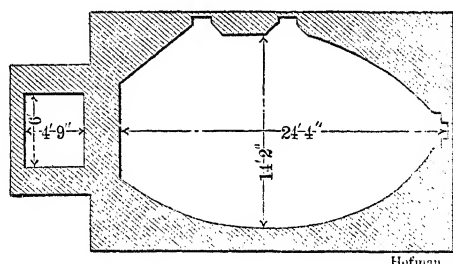
TABLE VI.—*Analyses of Flue-Dust and Briquettes.*

Flue-Dust from	Cu.				Fe.	Al <sub>2</sub> O <sub>3</sub> .	CaO.	S.		As and Sb.	Ag. Ton. Oz. per
	Total.	Sol. in H <sub>2</sub> O.	Sol. in H <sub>2</sub> SO <sub>4</sub> .	SiO <sub>2</sub> .				Total.	Sol. in HCl.		
Flue Near											
MacDougall furnace.	13.3	.....	.....	47.9	11.9	.....	.....	12.0	.....	.....	4.2
	11.8	.....	.....	52.0	11.0	.....	.....	7.9	.....	.....	4.8
Blast-furnace.....	10.6	.....	.....	33.2	18.7	.....	.....	10.7	.....	.....	5.3
“ “ .....	10.4	4.2	5.7	26.5	13.9	37.7	4.2	7.3	5.3	1.6	.....
“ “ .....	12.85	.....	.....	34.8	16.3	.....	.....	13.3	.....	.....	.....
Dust-chamber .....	13.3	1.6	5.4	31.7	8.3	15.8	0.5	7.6	.....	9.1	.....
“ “ .....	7.2	.....	.....	34.3	11.9	18.3	.....	7.27	.....	.....	3.4
Stack.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	14.3	.....
	6.3	.....	.....	40.0	20.8	.....	.....	9.3	.....	.....	54.0
Briquettes (dust and slimes).....	6.5	.....	.....	63.0	13.0	5.3	.....	8.0	.....	.....	.....
Briquettes (dust).....	12.5	MgO. 0.8	H <sub>2</sub> O. 13.05	36.0	13.9	11.0	2.7	8.96	Insol. 45.6	0.66	5.0

original Welsh furnace (Fig. 1), a radical change was then inaugurated by making the hearth oblong, tapering very quickly toward the bridge, but less so toward the flue (Fig. 2). The result was an enlarged hearth-area served by the same grate-area which, on account of the free development of the flame, could heat more effectively and smelt a larger amount of charge

per unit of fuel-consumption than with the old form. While at Argo, Col.,\* the reverberatory matting-furnace of 1878 had a hearth-area of 105 sq. ft. with a ratio of hearth-to-grate-area as 4.66 : 1, in 1894 the hearth-area had been increased to 481 square feet and the ratio to grate-area increased to 15.03 : 1.

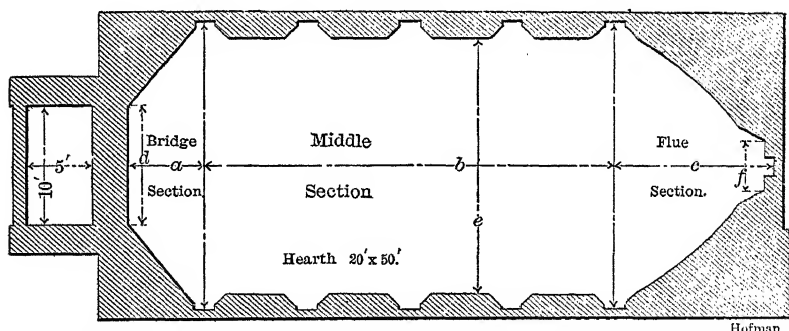
FIG. 1.



Horizontal Section of Welsh Type of Matting-Furnace, Used in Montana before 1891.

At present the Montana furnaces reach in round figures a hearth-area of 880 sq. ft., with hearths 50 feet long and 20 feet wide (in the middle section, Fig. 2); the ratio of hearth- to grate-area differs, however, only slightly from the Argo standard of 1894. Such a furnace treats 105 tons of charge in 24 h., producing

FIG. 2.



Horizontal Section of Improved Matting-Furnace, Used in Montana after 1891.

50-per cent. matte with a concentration of from 3 to 4 into 1. These large furnaces have many advantages over the smaller ones besides the greater capacity. The sides are less readily corroded by the slag (claying every 12 days versus every 3 days), as for a given quantity of matte less surface is exposed

\* Peters, *Modern Copper-Smelting*, 1895, p. 445.

TABLE VII.—*Examples of Montana Reverberatory-Furnace Smelting.*

	A.	B.	C.	D.	E.	F.
Length of hearth.....	50 ft.	50 ft.	49 ft. 6 in.	50 ft.	42 ft. 6 in.	37 ft.
Length, a, of bridge-section*.....	6 ft.	6 ft.	4 ft. 6 in.	6 ft.	42 ft. 6 in.†	.....
Length, b, of middle-section*.....	28 ft.	32 ft.	32 ft.	32 ft.	42 ft. 6 in.†	6 ft.
Length, c, of flue-section*.....	16 ft.	12 ft.	13 ft. 0 in.	12 ft.	42 ft. 6 in.†	.....
Width, d, of hearth at bridge*.....	10 ft.	10 ft.	10 ft. 9 in.	10 ft. 9 in.	15 ft. 9 in.	8 ft. 0 in.
Width, e, of hearth at middle*.....	20 ft.	20 ft.	20 ft. 2 in.	20 ft.	15 ft. 9 in.	14 ft. 6 in.
Width, f, of hearth at flue*.....	4 ft.	6 ft.	4 ft.	4 ft.	15 ft. 9 in.	5 ft. 0 in.
Hearth-area, sq. ft.....	842	886	878	878	688	.....
Hearth, thickness.....	22 in.	24 in.	19 in.	20 in.	28 in.	22 in.
Length of grate.....	10 ft.	10 ft.	7 ft.	10 ft. 9 in.	8 ft.	8 ft.
Width of grate.....	5 ft. 5½ in.	5 ft. 4 in.	5 ft. 6 in.	5 ft.	6 ft.	6 ft.
Depth of grate below top of bridge at bridge.....	1 ft. 10 in.	2 ft. 9½ in.	2 ft. 4 in.	2 ft. 8 in.	3 ft. 0 in.	.....
Depth of grate at opposite end.....	1 ft. 2 in.	2 ft. 6½ in.	1 ft. 4 in.	2 ft. 2 in.	.....	.....
Grate-area, sq. ft.....	53.9	53.3	53.0	53.75	Regenerative Gas Chambers	48.0
Ratio, hearth : grate-area.....	16.6	16.6	16.5	16.3	Reduction Chamber	.....
Height of roof above bridge.....	2 ft. 7 in.	2 ft. 6½ in.	2 ft. 10 in.	2 ft. 8½ in.	24 in.	2 ft. 0 in.
Height of roof above hearth at bridge.....	4 ft. 8 in.	4 ft. 4 in.	4 ft. 4 in.	4 ft. 8 in.	7 ft. 0 in.	4 ft. 10 in.
Height of roof above hearth at flue.....	2 ft. 1½ in.	2 ft. 10 in.	3 ft. 0 in.	2 ft. 2 in.	.....	2 ft. 4 in. to 2 ft. 8 in.
Width of bridge.....	3 ft. 0 in.	3 ft. 5½ in.	3 ft. 0 in.	3 ft. 0 in.	4 ft. 1½ in.	.....
Size of flue at vulcuary.....	6 ft. x 30 in.	6 ft. x 30 in.	5 ft. x 30 in.	6 ft. x 30 in.	.....	36 in. x 6 ft.
Size of flue leading to chimney.....	30 in. x 48 in.	30 in. x 30 in.	28 in. x 30 in.	30 in. x 30 in.	42 in. diam. at each end.	6 in. to 7 in.
Chimney, inside diameter.....	5 ft. 6 in.	6 ft. 3 in.	6 ft. 0 in.	6 ft. 1 in.	.....	30 in. x 36 in.
Chimney, height.....	70 ft.	.....	70 ft.	75 ft. 3½ in.	.....	4 ft. 6 in. x 4 ft. 6 in.

\* See Fig. 2.

† The hearth has the form of a rectangle.

TABLE VII.—*Examples of Montana Reverberatory-Furnace Smelting.*—(Continued).

	A.	B.	C.	D.	E.	F.
Charge, weight, tons.....	25	15	24	18	35	13
Charge, time of melting, hours.....	5	3 $\frac{1}{2}$	6 $\frac{1}{2}$	4 $\frac{3}{4}$	6	5 $\frac{1}{2}$
Charge, tons in 24 hours.....	112.5	105	90	90	150	70
Charge, tons per sq. ft. of hearth in 24 hours.....	0.129	0.118	.102	0.102	0.218	.....
Rate of concentration.....	5.7:1	4.7:1	7.56:1	5:1	3 $\frac{1}{2}$ :1	5:1
Fuel, bituminous coal, manner of firing.....	Direct.	Direct.	Direct.	Direct.	Prod. gas.	Direct.
Per cent. of ash.....	8.9	15.0	14.85	5.0	17.0-30.0	7-10
" " fixed carbon.....	51.9	44.5	44.5	55.0	45.0	50-52
Tons charge: 1 ton coal.....	3.05	3:1	4:1.	2.81:1	2:1	2.5:1
Labor, in 8-hour shift*.....	2+2	2+2	2+2 $\frac{1}{2}$	2 $\frac{3}{4}$ +1 $\frac{1}{2}$ †	†	2 $\frac{1}{2}$ +1 $\frac{1}{2}$
Matte, Cu.....	53.8	48.3	50	50	50	55
Ag. oz. per ton.....	45	35	100	66	20	.....
An. \$ per ton.....	.....	.....	.....	2(?)	.....	.....
Spec. gr.....	.....	4.8	.....	.....	4.77	.....
SiO <sub>2</sub> .....	36.8	42.8	34.5	38.6	41.9	49
Slag, Fe(Mn)O.....	51.9	47.31	48.0	51.4	42.7	32-36
Al <sub>2</sub> O <sub>3</sub> .....	8.4	7.5-8.0	8.5	1.5	10.9	.....
ZnO.....	.....	.....	14.0	.....	.....	.....
CaO.....	1.1	1.2	.....	3.3	1.1	.....
Cu.....	0.75	0.40	0.60	0.40	0.58	0.65
Ag.....	0.45	0.30	1.0	0.40	.....	.....
Sp. gr.....	.....	3.58	.....	.....	3.54	.....

\* The first figure represents the regular crew, the second the auxiliary labor.

† Not comparable in this manner on account of gas-producers.



to the slag; there is less variation in temperature on account of the greater reservoir of heat, which lengthens the life of the brick; the layer of slag is thinner, which allows quicker heating and causes less foaming when the charge is dropped from the hoppers; and the level of the matte remains more constant.

### *Fuel.*

The fuel used is bituminous coal of various grades from Montana, Wyoming and Utah, some of which are represented in Table VIII.

TABLE VIII.—*Average Coal-Analyses.*

Further analyses of coals are given in the Appendix (p. 316).

Locality and State.	Name of Mine.	Kind of Coal.	Years Averaged.	H <sub>2</sub> O. Per cent.	Volatile Matter.	Fixed Carbon.	Ash.
Belt, Mont.....	Belt.	Slack.	1898-'99-1902.	1.6	30.4	45.7	22.8
		Washed slack.	1899-1900.	0.8	33.0	49.2	17.0
		No. 1 nut.	1900.	0.2	38.1	47.2	14.5
		Pea	"	0.8	33.4	48.9	16.9
		Washed pea.	"	0.9	35.3	50.5	13.8
		Washed nut.	1901-1902.	1.3	29.2	50.2	19.3
		Lump.	1900-1901.	1.5	29.3	50.8	18.4
		"	(?)	2.2	24.25	46.65	26.9
		"	(')	2.4	22.1	43.2	32.2
		Mine-run.	1901.	0.9	27.6	47.3	24.2
Stockett, Mont.....	Sand Coulee.	"	(?)	3.9	24.9	49.6	24.6*
		Slack.	1898-1900.	1.2	30.8	46.1	21.9
		Washed nut.	1898-1901.	1.4	31.8	50.4	16.4
		Mine-run.	1899.	1.1	28.1	46.2	24.6
		Nut.	1898-1900.	1.1	30.3	45.8	22.8
		Lump.	1900-1901.	0.9	30.4	50.2	18.5
		Slack.	1898-1901.	1.1	30.1	44.3	24.5
		Nut.	"	1.1	29.7	44.7	24.5
		Lump.	1899-1901.	0.8	30.4	46.3	22.5
Sand Coulee, Mont.	Lochray.	"	"	1.6	30.2	47.3	20.9
		Mine-run.	"	1.0	30.0	47.7	21.3
	Gerber.	"	"	1.8	27.4	52.1	19.4
		Lump.	1900-1901.	0.9	31.2	48.9	19.0
	" "	Mine-run.	1899.	1.2	31.5	48.5	18.8
		"	"	1.0	28.5	56.7	17.7
	Millard.	Slack.	1899.	0.5	33.2	45.6	20.7
		Mine-run.	1899-1900.	0.8	31.1	46.7	21.4
	Lewis.	Slack.	1898-1899.	1.2	31.7	45.9	21.2
		Lump.	1898-1901.	0.9	30.4	49.0	19.7
Wyoming .....	Sweet Water.	Mine-run.	1899-1900.	1.0	30.5	46.6	21.9
		"	1902.	4.6	32.0	56.4	7.0
		"	"	5.4	31.8	50.4	12.0
		"	"	7.85	36.45	46.65	9.05
		"	"	2.6	37.58	45.12	14.70
Utah.....	Diamondville.	Slack.	"	4.7	36.52	44.56	14.85
		Lump.	"	4.8	38.0	59.1	1.9
		Gas-Coal.†	"	2.5	31.5	43.9	22.1
British Columbia..	Lethbridge.	Slack.	1898-1901.	4.6	31.6	45.0	18.8
		Nut.	1899.	2.9	34.6	41.2	21.3
Pennsylvania.....	Anthracite.	"	1900-1901.	0.5	7.8	80.6	11.1

\* Average of ten months.

† S 1.2 per cent.

Most furnaces are direct-fired. Some furnaces are worked with natural draught, but the majority have under-grate blast. In one instance the original fire-box has been changed to receive

a Duff (Pittsburg) gas-producer, with most satisfactory results as to saving of fuel and to shortening of the time required for melting. This latter is due to the fact that the ordinary grate has to be cleaned every four hours, while the grate of the Duff producer requires attention only once in 24 h. On account of structural difficulties the producer had to be given up. In many furnaces air is admitted through the roof above the fire-bridge in order to secure a better combustion and thereby save fuel. The air\* travels through the vault underneath the furnace, rises some distance in two corners of the stack (square on the outside, circular on the inside), descends in the others, returns underneath the hearth through 3 or 4 horizontal channels on either side of the vault, rises in corresponding vertical boxes near the bridge, enters both sides of an air-chamber on the roof above the bridge and descends through slots into the fire-bridge flue. There is some doubt prevalent as to the efficiency of this superheating. With most furnaces the heat of the flame passing out of the furnace goes to waste. According to Hixon† attempts had been made to join the Brown-Allen-O'Hara roasting-furnace to a reverberatory matting-furnace, but without success, owing to mechanical difficulties. Hixon suggests improvements which are to overcome them. Lang‡ patented a combination of reverberatory matting-furnace with a hand-reverberatory roaster; at Kedabeg, in the Caucasus, such a combination is said to be§ in successful operation. Attempts have been made by Thum|| to combine the Belgian zinc-smelting furnace with a hand-reverberatory roasting-furnace for roasting blende, but without success. The main difficulty would seem to lie in the fact that, in order to roast successfully in a reverberatory-furnace, it is essential to have a steady even temperature at the fire-bridge, and with a reverberatory-matting furnace, charged and discharged at intervals, this is not easily obtained, if it can be managed at all. In order to utilize the waste heat, one Montana plant has at-

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\* See, also, Peters, *op. cit.*, p. 456.

† *Notes on Lead- and Copper-Smelting*, New York, 1897, p. 59.

‡ *Engineering and Mining Journal*, August 6, 1898.

§ *Berg- und Hüttenmännische Zeitung*, 1891, 449; *The Mineral Industry*, vi., 247.

|| *Berg- und Hüttenmännische Zeitung*, 1874, 277.

tached a 300-H.P. Stirling boiler, Class A,  $21\frac{1}{2}$ , with 3,019 sq. ft. heating-surface to one of its 50- by 20-ft. furnaces. While an attempt of this kind in an Eastern refinery had proved a failure, owing to the diminished melting-power caused by the reduced force of the draught, here it proved an unequivocal success. At first, when the boiler was set up in the usual manner, the smelting-power of the furnace was reduced to an unreasonable amount, which was later corrected by removing nearly all of the baffle-plates and thus giving the gases a straight course through the boiler.

One plant only has reverberatory matting-furnaces with regenerative chambers, similar to the open-hearth steel-furnaces with stationary hearths. Tilting-furnaces (hearths 13 by 16 ft. with 176 sq. ft. area), similar to the Campbell open-hearth steel-furnace,\* which were built in 1890 and 1892, were abandoned on account of their small capacity and because of the fact that it was impossible, in pouring off the slag, to control the flow in such a manner as to prevent matte from passing over. Small quantities of matte in a large mass of slag cannot be distinguished by the eye. These furnaces were run, however, for several years as stationary furnaces, being gradually replaced by furnaces of larger size. The gas-producer used is the Evans-Klepetko;† it is a modification of the well-known Taylor producer, is 15 ft. high, has a 7-ft. bosh and a throat 7 ft. 4 in. in diam. The points in which it differs from the Taylor producer are a revolving top, a mechanical feed discharging into a hopper in the center of the top, a steam-jet reaching into the hopper which, with a pressure equal to that of the gas in the furnace, prevents this gas rising through the coal; 4 water-cooled stirring-arms attached to the revolving top which reach down into the bosh and are so distributed that they cover the entire surface during one revolution; lastly, an attachment for the sluicing of ashes as they are discharged by the revolving grate. The producer gasifies from 10 to 12.5 tons coal in 24 h., the top making one revolution in 3 minutes. The analyses in Table IX. give a comparison of the two kinds of coal used and of the ashes formed.

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\* *Trans.*, xxii., 360.

† *U. S. Patent*, No. 704,527, July 15, 1902.

TABLE IX.—*Analyses of Producer-Coal and Ash.*

MINE-RUN OF THE A. C. M. CO. BELT, MONT.								
Coal.*					Ash.†			
Total H <sub>2</sub> O.	Comb. H <sub>2</sub> O.	Volatile Matter.	Fixed Carbon.	Ash.	H <sub>2</sub> O.	Volatile Matter.	Fixed Carbon.	Ash.
4.0	1.0	23.6	51.3	24.1	1.1	10.4	45.4	43.1
4.7	1.8	21.4	48.4	28.4	1.4	3.3	34.3	61.0
4.4	1.4	24.4	49.3	24.9	1.5	4.3	30.4	63.0
4.1	1.4	25.4	48.8	24.4	0.9	12.3	40.5	46.3
4.2	1.4	23.9	46.9	27.8	1.6	4.6	36.3	57.5
4.1	1.4	25.5	49.9	33.2	1.2	9.6	24.1	70.1
MINE-RUN OF LOCHRAY MINE, SAND COULEE, MONT.*								
5.0	0.9	27.5	51.7	19.9	0.3	2.4	36.4	60.9
4.4	1.1	26.3	52.6	20.0	0.2	3.0	13.9	82.9
4.5	0.8	27.0	50.7	21.5	0.3	3.2	24.3	72.2
3.8	0.7	27.3	52.7	19.5	0.3	3.8	24.1	71.8
MINE-RUN OF GERBER AND LOCHRAY } MINES.....					0.3	7.0	32.5	60.3

\* For heat-values of these and of other coals, see Appendix.

† Not to be depended upon too much on account of difficulty of obtaining a good average sample.

The Anaconda coal alone does not work well in the producer; it is added to the Lochray coal, which alone gives very satisfactory results.

The ash from the Gerber-Lochray coal, freed from carbon, shows the following composition in percentages:

SiO<sub>2</sub>, 44.15; Fe<sub>2</sub>O<sub>3</sub>, 16.6; Al<sub>2</sub>O<sub>3</sub>, 37.9; CaO, 1.0; MgO, 0.4; S, 0.04.

### *Details of Practice.*

The coal is crushed through a  $\frac{3}{4}$ -in. ring and passed through a 5-mm. screen; the oversize goes to the producers; the fines, forming about 30 per cent., are burnt under the boilers. It will be seen that, owing to the high percentage of ash in the coal, the amount of fixed carbon retained by it is exceedingly high. The flue from the 6 producers serving one matting-furnace has to be fired once in 8 days to remove soot, tar, etc., and

washed out once in 6 weeks to free it from ashes. An average analysis of producer-gas, taken from many determinations, gave in volume-percentage:  $\text{CO}_2$ , 9.4;  $\text{CO}$ , 16.8;  $\text{CH}_4$ , 2.9;  $\text{H}$ , 13.3;  $\text{N}$ , 57.6. The high percentage of carbon dioxide is probably due to the excessive use of steam. Bunte\* has shown that the larger the amount of steam that is admitted to a producer the larger is the volume of carbon dioxide formed. The gases, however, pass off from the producers with a temperature of  $700^\circ \text{C}$ ., which would seem to counteract the cooling effect of the steam and cause the carbon to burn rather to carbon-monoxide than to dioxide.

The hearth of the regenerative matting-furnace, 42 ft. 6 in. by 15 ft. 9 in., has the same width at the ends as at the middle. While this arrangement gives a larger hearth-area than in the furnace tapering at the ends, it has the disadvantage that the sides tend to bulge inwardly instead of outwardly as is ordinarily the case. The furnace is built in 5 sections in order to allow for expansion, which amounts to 14 in. over all. The sections are separate in the roof; in the 18-in. side-walls the necessary spaces are left between the bricks. The upper tie-rods pass 1 ft. above the roof (from 9 to 12 in. thick), as this may rise as much as 9 in.; the lower tie-rods, placed 4 ft. below the brick hearth supporting the quartz-bottom, were passed at first through 4-in. wrought-iron tubes; when these burnt out, the new tie-rods were provided with a small central opening for water-cooling. At present, the rods are solid,  $1\frac{1}{2}$  in. in diam., and pass through a flue 2 ft. 6 in. by 3 ft. 8 in., which is ample for air-cooling. The checkerwork has a peculiar construction to permit cleaning and thus reduce the slagging of the brick through dust. The flues are horizontal, 18 in. high by 7 in. wide and from 11 to 21 ft. long, according to position; they are made of fire-brick. The 9-in. brick forming the tops and bottoms are laid  $4\frac{1}{2}$  in. apart, leaving open spaces to allow the dust to fall and collect in a pit at the bottom, from 4 to 5 ft. deep. The heating-surface is not very much diminished by this arrangement, as the sides of the horizontal bricks are exposed to the hot gases; this would not be the case if the bottoms and tops did not have the open spaces. The dimensions

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\* Ledebur, *Gasfeuerungen*, Leipsic, 1891, p. 34.

of the air-chambers are;—width 8 ft. 6 in., length 14 ft.  $7\frac{1}{2}$  in., height 13 ft. 9 in.; of the gas-chambers, width 7 ft., length 14 ft.  $7\frac{1}{2}$  in., height 13 ft. 9 in. The heating-surfaces, including exposed walls, are 4900 and 4200 sq. ft. respectively. While long-checkers save fuel, they tend to shorten campaigns; the reverse is the case with the short-checkers. Air enters the hearth through a single port, 15 ft. 9 in. by 2 ft., the gas through 4 ports,  $20\frac{1}{4}$  in. by 30 in. The products of combustion from the furnace descend in a chamber at one end of the checkers, and, after passing through these, ascend in the other at the opposite end. The life of the checkers is about 3 months. In order to lengthen it, experiments are under way to have the gases go, during the charging-period, through a by-pass direct into the main flue, and thus prevent the inevitable dust from settling in the checker-flues.

The working-bottoms of the furnaces, as shown in the table, have a thickness ranging from 19 to 28 in. at the lowest point, the tap-hole. Both single- and double-bottoms are found. The single-bottom is more common with ore-smelting, as here the furnace had to carry a relatively light material in comparison with the refining-furnace in which the double-bottom is more prevalent. The bottom-sand is a sugar-like quartzite, with 95 per cent. silica, the rest being alumina and ferric oxide. It is crushed between rolls to bean-size, say, through a  $\frac{1}{4}$ -in. ring; 80 per cent. of it will pass through a 5-mm. screen. Two examples of putting in a bottom will show how great are the differences in practice. One furnace has a hearth 20 by 50 ft., is direct fired, and takes 73 tons of crushed sandstone. Supposing the furnace to be dry and warm, from 6 to 7 tons of sand are charged and heated for 6 h., being rabbled at intervals. These operations are continued until one-half of the total sand required has been introduced, when the whole is heated to the fritting of the surface. The other half of the sand is now charged in the same manner as was the first, and brought to fritting. Converter-slag is now poured in to cement the particles of sand more firmly together; from 3 to 4 charges of calcines are worked, and the furnace tapped dry after the fourth charge. The furnace is allowed to cool for 3 or 4 h. and is ready for regular work. The time required for putting in the bottom is 4 days. The bottom under consideration was

put in in February, 1900, and shows little wear to-day (June, 1902).

The other extreme is represented by a gas-fired furnace with regenerative-chambers; the rectangular hearth is 42 ft. 6 in. by 15 ft. 9 in. It is presupposed that the furnace is new and requires drying and warming. A light wood-fire is started on the hearth in half a dozen places, the valves are reversed every half-hour. In 10 days the furnace will be at a dull-red, in 2 days more it will be hot enough to ignite producer-gas. The gas is turned on, in small amount at first; after 3 days the wood-ashes are taken out; on the 4th or 5th day (the 16th or 17th after starting) the furnace will be a bright cherry-red. Five tons of sand are charged through the hoppers in the roof and spread, covering the hearth to a thickness of 2 in. The sand is calcined, turned over, and the furnace brought to a normal heat; charging of sand and heating it are continued in 5 or 6 h. intervals until about 60 tons have been introduced, care being taken to have the sand thoroughly calcined. The furnace is now brought in from 36 to 48 h. to the highest heat obtainable without endangering the brickwork, and the sand brought to a set which amounts to about 3 in. in thickness. The high heat is maintained for 16 h., and the furnace then cooled down in 8 h. to a dull cherry-red. Enough calcines (giving 45-per cent. copper-matte) are dropped through the roof to cover the hearth to a depth of 3 in.; they are melted down and are absorbed by the sand-bottom. The furnace is cooled as before, and a second charge of calcines given and melted. Part only of the charge will be taken up by the hearth, the rest is tapped out. The furnace is again cooled, from 12 to 15 tons ore-charge are given, melted down, and the furnace tapped dry and cooled to a dark-red, but more slowly than before. It is again fired up, worked for 3 days as under normal conditions, and then tapped dry and allowed to cool to a cherry-red. It is now safe for any kind of work. The time required to bring a new, cold furnace to this point is 3 weeks. The object of the repeated coolings of the furnace is to harden the bottom. A bottom put down without cooling is liable to be mushy when it is hot, even though it be hard when relatively cool.

The ore-charges consist mainly of roasted concentrates, with

finer from first-class ore; sometimes small additions of limestone are made. In converter-plants, low-grade blast-furnace matte is often poured into the reverberatory matting-furnace in order to raise the percentage of copper.

In regular work, the charge is collected above the furnace in from 5 to 7 hoppers, having discharge-pipes with gates, and dropped onto the hearth. With non-regenerative furnaces, the bulk of the charge is collected near the fire-bridge; thus with 5 hoppers, *e.g.*,  $7\frac{1}{2}$  volumes of the charge will be dropped through hopper No. 1 near the fire-bridge, then  $6\frac{1}{2}$  volumes through hopper No. 2, 5 through No. 3, 3 through No. 4, and 1 through No. 5 near the flue-bridge. With a regenerative-furnace having, for example, 7 hoppers, charge will be first dropped through the end-hoppers that it may flow toward the center, then through the central hopper and its 2 neighbors, and lastly through the 2 remaining ones, the flow being regulated with the gates that there may be an even distribution. The charge is now melted down and stirred at intervals, in order to break up lumps. With a furnace that has been tapped dry, the bottom has to be rabbled in order to bring adhering melted parts to the surface. When melted and thoroughly liquefied, the slag is skimmed into a settling-box or a Devereux-Livingstone settling-pot,\* that particles of matte carried out by the slag may be collected and saved. With zincose ore it is important to give a good fire before tapping, that there may be a good separation of the mushy, zinky matte (which being light, floats on the surface of the heavy matte), and the slag. The overflow-slag is either collected in waste-slag pots of from 5 to 6 tons capacity, hauled to the dump electrically and poured, or granulated and carried off by a stream of water. Granulated slag from the reverberatory matting-furnaces was first introduced in 1888 at the Anaconda Lower Works. In order to reduce the copper-contents of this slag, fine pyrites has been sprinkled into it before the final firing-up, with the hope that slagged copper may be sulphurized and carried into the matte. The matte is removed from the furnace wholly or only in part, depending on whether it is to be collected and shipped or transferred to the converters. When the

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\* Hofman, *Metallurgy of Lead*, 1899, p. 267.



furnace is to be tapped dry, which occurs only after several charges (with high concentrations as many as 12 charges) have been treated, the hearth holds from 18 to 20 in. of matte which is run into sand-moulds or cast-iron moulds of similar form, or it is first collected in a steel-ladle, and then poured into the form of flat cakes. With a converter-plant only part of the matte is removed after every charge, the aim being to keep the level as nearly constant as possible. With furnaces that are tapped dry, the sides are patched or fettled after every tap. With converter-plants, fettling of the matting-reverberatories is resorted to only once every 4 or 5 weeks. As fettling-material, siliceous gold-ore has, in some instances, replaced the usual sand. A furnace runs about 90 days before it needs sufficient repairs to require shutting down. The subjoined partial analysis of flue-dust collected in the gas-checkers of a furnace gave:

Insoluble, 91.1;  $\text{SiO}_2$ , 62.2;  $\text{Fe}_2\text{O}_3$ , 8.2;  $\text{Al}_2\text{O}_3$ , 25.7;  $\text{CaO}$ , 1.8;  $\text{MgO}$ , none; S, 0.30;  $\text{CuO}$ , 2.3 per cent.

Evidently, considerable coal-ash was carried over by the gas.

The material formed in the air-checkers resembles slag very much in analysis, except that it is higher (*i.e.* from 5 to 9 per cent.) in copper, silica and alumina, and a trifle lower in iron.

## VII. CONVERTING COPPER-MATTE.

The converting of copper-matte has undergone many changes in the form of apparatus and the mode of working since 1884, when the first converter was put into operation.

The original Parrot converter,\* swinging in a vertical plane, was  $8\frac{1}{2}$  ft. high and 5 ft. in diam., outside dimensions. The matte was melted down in a cupola and tapped into the converter through a launder. In 1886, the original method of working in separate stages was changed, and 50-per cent. matte was brought forward to coarse copper in one operation. As the initial charge was 2,500 lb. of matte, increasing to a maximum of 9,000 lb., the siliceous lining being eaten and worn away, the amount of coarse copper produced in a blow was relatively small. The largest upright converter in Montana† is

\* Peters, *op. cit.*, p. 529. Stickney, *The Mineral Industry*, i., 154.

† The largest upright converters are those of Aguas Calientes, Mexico: height, 16 ft.; diameter, 8 ft.

14 ft. 2½ in. in height and 7 ft. in diam., outside dimensions. It works with an average charge of 5 tons and a final charge of 11 tons of 50-per cent. copper-matte, receiving it as direct matte from the forehearth of the blast-furnace or from the reverberatory-furnace. Working direct matte was planned in 1890-91 by the late C. O. Parsons, and carried out by F. Klepetko in 1892; matte was transferred from reverberatory- and blast-furnaces by means of ladle and electric crane to the converter. At two plants a medium-sized converter was lifted by means of an overhead traveling-crane from its trunnion supports and brought to the blast- or reverberatory matting-furnace to receive its liquid-charge; this method has become obsolete, but it has its advantages.

At present, the Parsons-Klepetko method is generally employed. A converter-charge is tapped from the blast-furnace forehearth, or the reverberatory matting-furnace, into a steel ladle managed by an electric traveling-crane and poured into the converter. Ladles vary in size; a common form has a diam. of 5 ft. and a height of 3 ft. 10 in. The lining, ordinary loam, is plastered on by hand, dried by an air-current and then by a fire (wood, coke); converter-slag, to the thickness of 3 in., is then poured in. Matte-ladles are used, also, as slag-ladles. A lining is good for 9½ tons of copper.

When the 50-per cent. matte has been blown to white metal, the slag is skimmed and the converter very often recharged with 50-per cent. matte and blown through the slagging-stage, but, more commonly, the original charge of matte is blown to coarse copper after skimming, without addition of fresh matte. Doubling charges is a common procedure on matte running less than 50 per cent. of copper. After skimming the slag, the white metal is brought forward to coarse copper. By these double-charges a considerable amount of copper is produced in one blow, to be poured into the form of shipping-ingots, of from 200 to 250 lb. in weight, or anodes, or into a refining-furnace, to be refined before casting into anodes.

The upright converter has been replaced in the new plants by the David-Manhès, or barrel- or trough-converter swinging in a horizontal plane. Here we find two sizes; the smaller, in more general use, is 7 ft. in diam. and 10 ft. 6 in. long; it receives an initial double-charge of 5 tons and a final double-

charge of 12 tons of 50-per cent. matte: the larger one is 8 ft. in diam. and 12 ft. 5 in. long, the initial double-charge is 10 tons, and the final double-charge is 20 tons. The main reason for replacing the upright by the horizontal converter is the low blast-pressure required by the latter type, 10 vs. 15 lb., although this advantage suffers greatly by the difficulty of making the lining firm; a thing relatively easily accomplished in the vertical converter. Converting matte in a bottom-blown upright vessel has been tried. This vessel was 13 ft. high and 7 ft. in diam., had sixteen  $\frac{3}{4}$ -inch tuyeres placed in a manner similar to that which obtains in making steel by the Bessemer process. The experiments were successful so far as the process proper was concerned, *i.e.*, 50-per cent. matte was satisfactorily brought forward to coarse copper, which remained sufficiently fluid to be poured satisfactorily. They appear to show that the failure of Manhès to use a bottom-blast converter\* in his first experiments was probably due to his operating with too small a vessel from which matte and slag were blown out, and with too little matte, which furnished only a small amount of copper quickly chilled by the blast. The Montana experiments have not been followed up for the present, as the life of the bottom was too short, parts of it floating up during a blow. The experience gained so far tends to show that the bottom-blown converter works more quickly than the side-blown, that the wear of the lining is more even, and that the critical points are less easily discovered and have to be watched with great care. When the question of lining the bottom shall have been settled satisfactorily, it is believed that blowing from the side will be given up, at least with a large converter having an initial charge of 5 tons 50-per cent. matte. A gyratory motion of the charge, obtained by placing the bottom-tuyeres inclined, as in the David "*Selecteur*,"† may lengthen the life of a converter-lining.

In all plants the converters are placed in a single row, forming a straight line. Back of them is the main flue (vacuum 0.1 in. water), with projecting hoods, into which the converters discharge their gases, vapors and finely divided particles of

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\* Grüner, *Bulletin de l'Industrie Minérale de Ste. Étienne*, 1885, xiv., 607.

† P. L. Burthe, *Annales des Mines*, 1898, xiii., 621. *Engineering and Mining Journal*, October 22, 1898.

charge (called flue-dust). The main flue ends in a dust-chamber (temperature,  $370^{\circ}\text{C}.$ ), connected with a stack. At some works the matte is poured into a converter at the front from a suspended ladle, at others it arrives at the back (above the main flue) in a ladle, and is poured into a launder, which delivers it at the front. A stall is served by from 2 to  $2\frac{1}{2}$  converters and from  $1\frac{1}{2}$  to 2 ladles. The lining of converters, both of body and of caps, is still causing trouble, as in the most favorable case the body has to be removed after 28 tons of copper have been treated; the cap has a longer life. The material is, of course, siliceous. A greenish granular sandstone ( $\text{SiO}_2$ , 64.4;  $\text{FeO}$ , 7.6;  $\text{Al}_2\text{O}_3$ , 16.5;  $\text{CaO}$ , 1.1;  $\text{MgO}$ , 1.8; Ignition-loss, 4.7), which readily crumbles when exposed to the air, is used without any admixture of a bond. A crushed quartzite, pugged with 5 per cent. clay, gives a satisfactory mixture ( $\text{SiO}_2$ , 88.6;  $\text{Fe}_2\text{O}_3$ , 3.1;  $\text{Al}_2\text{O}_3$ , 4.0;  $\text{CaO}$ , 1.6); lining-sand has been mixed with slimes from the ore-dressing works (Insoluble, 75.8;  $\text{SiO}_2$ , 60.0;  $\text{Fe}$ , 2.2;  $\text{Al}_2\text{O}_3$ , 19.9;  $\text{S}$ , 6.8;  $\text{As}$  and  $\text{Sb}$ , 0.75;  $\text{Cu}$ , 6.05) and with second-class ore (Insoluble, 68.9;  $\text{SiO}_2$ , 57.0;  $\text{Fe}$ , 9.6;  $\text{Al}_2\text{O}_3$ , 10.4;  $\text{S}$ , 15.95;  $\text{Cu}$ , 5.95;  $\text{Ag}$ , 2 oz. per ton), giving material of the following percentage composition:

Insoluble, 82.2;  $\text{SiO}_2$ , 63.0;  $\text{Fe}$ , 5.9;  $\text{Al}_2\text{O}_3$ , 17.3;  $\text{CaO}$ , 1.5;  
 $\text{MgO}$ , 1.3;  $\text{S}$ , 2.1;  $\text{Cu}$ , 1.0;  $\text{H}_2\text{O}$ , 11.0; Ignition-loss, 4.1.

Another mixture for the body of the converters consists of quartzite ( $\text{SiO}_2$ , 90 to 91;  $\text{Al}_2\text{O}_3$ , 2.0;  $\text{FeO}$ , 3.0;  $\text{Ag}$ , 10 to 20 oz. per ton) and slimes ( $\text{SiO}_2$ , 60;  $\text{Fe}$ , 4 to 5;  $\text{Al}_2\text{O}_3$ , 15;  $\text{S}$ , 9;  $\text{Cu}$ , 5 to 7), to form a lining ( $\text{SiO}_2$ , 85;  $\text{Fe}$ , 3.3;  $\text{Al}_2\text{O}_3$ , 4;  $\text{S}$ , 3;  $\text{Cu}$ , 4;  $\text{Ag}$ , 12 oz. per ton). Table X., on page 306, gives an analysis of all the materials used to make the converter-lining.

In some instances siliceous gold-ores have been worked in as a lining-material. The material for the cap is generally made more plastic than that of the body. The lining usually consists of body-lining material, and from 5 to 10 per cent of clay, mixed with more water than the body-material. If the lining of the body were eaten away solely by the scorifying action of the iron, its life would be much longer than it is. The chemical action in some instances plays even a subordinate part to the mechanical wear, by the swash of the liquid charge which causes part of the lining to break off. This is especially observable in the barrel-converter. The chemical wear takes

TABLE X.—*Analysis of Materials of Converter-Linings.*

	SiO <sub>2</sub> .	Fe.	Al <sub>2</sub> O <sub>3</sub> .	CaO.	MgO.	Loss on Ignition.	S.	Cu.	Ag.	As and Sb.	H <sub>2</sub> O.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Oz. P. T.	Per Cent.	Per Cent.
Greenish, granular sandstone, Great Falls.....	64.4	5.9	16.5	1.1	1.8	4.7	.....	.....	.....	.....	.....
Crushed quartzite, with 5 per cent. clay.....	88.6	2.2	4.0	1.6	.....	.....	.....	.....	.....	.....	.....
Sandstone, slimes and second-class ore.....	68.0	5.9	17.3	1.5	1.3	4.1	2.1	1.0	.....	.....	11.0
Quartzite and slimes.....	85.0	2.5	4.0	.....	.....	.....	3.0	4.0	12.0	.....	.....
Sandstone.....	64.4	5.9	16.5	1.1	1.8	4.7	.....	.....	.....	.....	.....
Slimes.....	60.0	2.2	19.9	.....	.....	.....	6.8	6.05	.....	0.75	.....
Second-class ore.....	57.0	9.6	10.4	.....	.....	.....	15.95	5.95	2.0	.....	.....
Quartzite.....	90 to 91	2.7	3 to 2	.....	.....	.....	.....	.....	10 to 20	.....	.....
Slimes.....	60.0	4 to 5	15	.....	.....	.....	9.0	5.7	.....	.....	.....

place mainly near the tuyeres, the mechanical wear at the opposite side near the top of the body. In an upright converter, 14 ft. 2 in. high and 7 ft. in diam., the bottom lining will be 22 in. thick, the side-lining at the tuyeres will be 30 in. thick, and on the opposite only 18 in., while at the top of the body the lining on the tuyere-side will be  $17\frac{3}{4}$  in. and opposite  $27\frac{3}{4}$  in. thick.

In a horizontal converter, 10 ft. 6 in. long and 7 ft. in diam., the bottom will be 18 in. thick, consisting of a 4-inch segmental layer of mixture tamped onto the steel shell, followed by a slightly curved  $4\frac{1}{2}$ -in. course of brick and by a 10-inch layer of mixture. The sides are lined with a course of  $4\frac{1}{2}$ -inch fire-brick, special brick  $6\frac{1}{2}$  by  $6\frac{1}{2}$  by 4 in. being used at the tuyere-level. A cross-section through the tuyere-level will show the sand-lining to be 30 in. at the tuyeres and 16 in. opposite; at the top of the body, the corresponding dimensions will be 20 and 20 in. The longitudinal section will show the side-lining to be 16 in. at top of the bottom-lining and 34 in. at the top of the body. The ends are not lined with brick. Sometimes the lining is rammed in by hand, still an air-drill adapted for the purpose or the flask-tamper of the foundryman is more common.

The ramming is done by contract, a premium being paid if the converter makes more than the standard amount of copper on a lining and deductions made if it falls below it. If the lin-

ing is to be patched, the vessel is first emptied and then cooled with water from the outside, then lining-material is rammed in or a few scoopfuls of infusible odds-and-ends are thrown in, tamped down and slag thrown over the patch. Occasionally, matte is run into the empty converter and allowed to solidify. Experiments in lengthening the life of the converter, by pouring siliceous blast-furnace slag into the converters with the hope that the excess of silica over that of the normal converter-slag would combine with the oxidized iron from the matte, have been only partially successful.

The difficulty and expense of handling converter-slag has been overcome in one plant by the use of a slag casting-machine.

The average time to complete a charge with a 5-ton upright converter is 2 h. 55 min.; this is reduced to 2 h. 20 min., if the time taken up by pouring and waiting is excluded. The blowing of a single-charge takes 47 min., a double-charge takes 82 min. In pouring the slag a rabble is pushed in and out of the stream of slag. The appearance of bright sizzling specks is a sign that white metal is coming over. A few partial analyses of the products obtained in converting are subjoined:

	Cu.	SiO <sub>2</sub> .	FeO.	Al <sub>2</sub> O <sub>3</sub> .	CaO.	Ag. Au. Oz. PerTon.	S.	As. and Sb.
Slag.....	2.0	30.0	55.2	9.5	1.0	.....	.....	.....
" .....	1.6	29.8	57.8	10.0	1.0	0.8	.....	.....
" .....	5-6	31-32	54	.....	.....	.....	.....	.....
Dust.....	37.8	1.5	7.4	1.9	None.	.....	14.3	4.8
" .....	63.4	3.8	12.8	.....	.....	53.8	16.0	.....
" .....	65	.....	.....	.....	.....	.....	.....	.....
Pig-copper .....	99.0	.....	.....	.....	.....	9-40	.....	.....

The labor required per shift for a single converter is made up of 1 skimmer, 1 puncher, 1 helper and 1 sampler.

With 3 large horizontal converters running as steadily as possible the number is 1 skimmer, 2 punchers, 2 helpers and 2 samplers.

In Table XI., on p. 308, a part of the data given previously has been summarized, and some additional facts have been added.

It will be seen that the medium barrel uses much less lining per ton of resulting copper than the upright vessel, although the grade of the matte differs only slightly. This may be ex-

TABLE XI.—*Montana Converters.*

Converter.	Grade of Matte.	Hor. Section at Tuyeres.	* Lining Material per Converter.	Lining per Ton Cop- per Resulting.	Copper per Converter.	Tuyeres.		Maximum Pressure.
	p.c. Cu.	Sq. Ft.	Tons.	Tons.	Tons.	No.	Diam.	Lb.
Upright vessel, 7 ft. x 14 ft. 7 in.....	51.1	7.07	7.63	.53	14.8	8	$\frac{3}{4}$	18
Medium barrel, 7 ft. x 10 ft. 6 in.....	51.7	6.66	8.40	.32	26.2	14	$\frac{3}{4}$	12
Large barrel, 8 ft. x 12 ft. 6 in.....	45	17.79	16.60	.80	20	18	1 $\frac{1}{8}$	15

plained by the more siliceous character of the lining used, as, except for the mechanical effect of making the material plastic, alumina would be of no service in scorifying; furthermore, the lining of the upright vessel runs higher in iron.

The lining of the large barrel is of about the same quality as that used with the medium barrel, but as the matte of the former is of a lower grade, the quantity of lining used per ton of resulting copper is naturally higher.

Converter Lining.	SiO <sub>2</sub> .	Fe <sub>2</sub> O <sub>3</sub> .	Al <sub>2</sub> O <sub>3</sub> .	CaO.	MgO.
Medium Barrel,	91.	3.	3.8	1.6	.....
Upright Vessel,	64.4	8.4	16.5	1.1	1.8

In the upright vessel, the amount of free air used per ton of copper converted is about 200,000 cu. ft., not allowing for "leakage" in the blowing engines.

#### *Yield of Metal.*

The general yield of metal in roasting, smelting, and converting, in treating an ore assaying from 10 to 12 per cent. copper and about 4 oz. silver per ton, is copper 92+ and silver 95 + per cent.

### VIII. ELECTROLYTIC REFINING OF COPPER.

The work of the two electrolytic refining-plants in the Butte district, at Great Falls and Anaconda, is of especial interest,

\* These data do not show the amount of lining-material required to line a new converter, but only the average weight put in after a lining is worn down to the point where further charges would endanger the iron shell.

as they work under different conditions of power, while they treat similar anode-copper by the multiple process. The works at Great Falls, driven by water-power, have 312 tanks, 9 ft. 9 in. long by 2 ft. 4 in. wide by 3 ft. 9 in. deep, in three sets of 104 tanks, each set having a separate solution; a tank is charged with 20 anodes and 20 cathodes; the current is of 40 amperes per square foot of cathode area; two Westinghouse direct-current machines generate 9000 amperes at 200 volts. The works of Anaconda,\* driven by steam-power, have 1,400 tanks, 8 ft. 2 in. long by 4 ft. 5 in. wide by 4 ft. 6 in. deep, in sets of 200 each. A tank is charged with 38 double anodes and 80 cathodes, the current is of 10 amperes per sq. ft. of cathode area. There are 7 dynamos, each generating 4,000 amperes at 60 volts for 200 vats.

The anodes of the Butte district assay Cu, from 98 to 99 per cent., Ag, from 40 to 120 oz. per ton, and Au, from 0.1 to 1.5 oz. per ton. They are cast either direct from the converter or the converter-copper is first refined in a reverberatory furnace.†

The converter-charge is blown to the point at which the copper still retains considerable sulphur dioxide in solution. In pouring anodes from the converter, the copper strikes a launder swung in position, which delivers the copper into open flat moulds resting on a car moved slowly on rails laid between the posts of a stand. The strong evolution of sulphurous gas upon cooling makes the anodes very uneven. The moulds are of copper, but have cast-iron centers that take up the impinging force of the stream of metal. If the copper is first to be refined, the metal from several converters is emptied into ladles which discharge their contents into the stationary refining-furnaces. Here the rabbling of the copper, in order to cause the oxidation necessary to produce set copper, has been replaced by forcing air into the copper by 1-in. wrought-iron pipes inserted through the side- and end-doors of the refining-furnace. The two hours required for rabbling have been reduced to one and one-quarter hours by blowing; six feet of pipe

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\* *Engineering and Mining Journal*, September 19, 1896.

† Refining converter-copper before casting into anodes has been given up at Great Falls in spite of the advantages mentioned later on in the text as regards corroding, because the gain did not warrant the expense. At Anaconda, where power is generated by steam, it is of considerable advantage to have refined anodes.



are used up with a 40-ton charge. Oxidation is continued until a small ladle-sample, when broken, shows the characteristic single bubble in the depressed center. Care is taken to bring the copper only to this point, when the copper is saturated with cuprous oxide, as otherwise the surface will quickly become oxidized and covered with melted oxide. The set copper is then poled to tough-pitch copper, 15 poles (3 in. at end, 7 in. at butt and from 15 to 18 ft. long) being required for 50 tons of copper. From the reverberatory-furnace the anodes are cast by means of an improved form of Walker casting-machine.\*

A tilting reverberatory-furnace constructed along the general lines of the Brückner cylinder was used at the old Anaconda works, but was given up after a short trial.

Converter-anodes are less evenly corroded, give more scrap and furnish a mud of lower-grade than do anodes from the refining-furnace. Coarse copper, *e.g.*, with Cu, 99.27 per cent.; As and Sb, 0.07 per cent.; Ag, 61.14 and Au, 0.22 oz. per ton, cast from the converters into anodes, gave 8 per cent. scrap, and the anode-mud, after having been passed through a 40-mesh sieve, washed and dried, contained Cu, 41 per cent.; Ag, 7000, and Au, 18 oz. per ton. Similar copper, refined in the reverberatory-furnace and then cast into anodes, gave in corroding only 5.5 per cent. scrap, while the 40-mesh anode-mud assayed Cu, 18 per cent.; Ag, 15,000, and Au, 38 oz. per ton.

The Great Falls anodes are 24 in. wide by 35 in. long by 3 in. thick at the top and  $2\frac{1}{2}$  in. thick at the bottom; they are beveled at the bottom; the two lugs on top are  $4\frac{1}{2}$  in. wide by  $2\frac{1}{2}$  in. long by 2 in. thick. The anodes are suspended from cast-copper cross-bars ( $1\frac{1}{2}$  in. by  $1\frac{1}{4}$  in.) by means of the Morrow clip,† a loop of sheet-copper,  $2\frac{1}{8}$  in. wide (recently replaced by a rod  $\frac{1}{4}$  in. in diam.) by  $3\frac{1}{2}$  in. high, the ends of which are allowed to project into the anode-moulds before they are filled with liquid-copper. A converter-anode weighs 500 lb., and is sufficiently corroded in about 18 days to be withdrawn and regarded as scrap, a new anode being put in its place. Anaconda anodes are from 24 to  $24\frac{1}{2}$  in. wide,  $32\frac{7}{8}$  in. long, and  $1\frac{1}{4}$  in. thick at the top, and 1 in. at the bottom. They have 2 top-

\* *The Mineral Industry*, vii., 252.

† *U. S. Patents*, No. 621,121, March 14, 1899; No. 631,471, August 22, 1899.

lugs 6 in. wide with holes, and are suspended in pairs from iron copper-covered cross-bars ( $\frac{1}{2}$  by  $1\frac{3}{4}$  in.), 19 to a tank, by means of copper-hooks  $\frac{3}{8}$  in. thick. An anode weighs 230 lb. and is corroded in 37 days.

The cathodes are made in the usual way, in tanks set apart for the purpose. They are slightly longer and wider than the anodes. At Great Falls they are 26 in. wide by  $36\frac{1}{2}$  in. long and weigh  $2\frac{1}{2}$  lb.; at Anaconda they are 11 in. wide by 33 in. long and weigh from  $\frac{3}{4}$  to 1 lb. At Great Falls the cathodes are suspended by the Morrow clip, which is fastened to the starting-sheet by a machine,\* which punches a hole through the loop and sheet, bends over the lugs and clamps them fast. At Anaconda the upper rim of the starting-sheet is bent and clamped over the ends of a piece of sheet-copper,  $4\frac{1}{2}$  in. wide by 11 in. long; 4 cathodes go to a cross-bar; the electrode-distance ranges from  $1\frac{1}{2}$  in. to 2 in.

The tanks are made of 3-in. plank and are lined with 8-lb. lead, the bottoms have a board-cover to protect the lead-lining from pieces of anode-copper that may become detached. At Great Falls, the vats are arranged in pairs with 3 conductor-bars, the central bar serving as cathode-bar for 1 tank, and as anode-bar for the other.† At Anaconda, with the Thofehn disposition, the tanks are arranged in single rows.‡

The electrolyte contains with the heavier current 170 grammes concentrated sulphuric acid and 42 grammes copper per liter; with the lighter, 150 grammes acid and 40 grammes copper. The resistance appears to begin to increase when the copper rises above 42 grammes. The temperatures of the electrolyte at the head tanks are  $64^{\circ}$  C. and  $50^{\circ}$  C., and the circulation, 6 and 3 gals. per minute. Solutions are admitted and drawn off, either at the ends or the sides, precautions being taken by means of perforated lead baffle-plates to have an even distribution, and thus avoid stirring-up any mud. The different forms of plunger-pumps, which were formerly used to raise the electrolyte, have given way to Pohle's air-lift pumps working under a pressure of 25 lb. These have proved very satisfactory, especially since the delivery-pipe has been cut in two near the discharge and the ends joined by a rubber hose, thus breaking any electric

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\* U. S. Patent, No. 600,498, March 8, 1898.

† Illustration in *The Mineral Industry*, viii., 186.

‡ *Berg- und Hüttenmännische Zeitung*, 1893, p. 54.

connection. The electrolyte becomes enriched in copper and takes up iron, arsenic and antimony. A foul solution contained, *e.g.*, per liter:

Cu, 51.8 grammes; Fe, 13.2 grammes; As, 14.02 grammes;  
Sb, 0.62 grammes;  $\text{H}_2\text{SO}_4$ , 48 grammes.

The low percentage of antimony is due to the practice of adding daily enough concentrated crude hydrochloric acid to the head tank to maintain 0.04 grammes chlorine per liter in solution. This addition precipitates antimony, it is believed, as oxychloride. When there is deficiency of hydrochloric acid, a sample-plate becomes streaked, tarnished, black and, at the same time, brittle. The excess of copper over the normal is removed by shunting off part of the electrolyte and making it pass through tanks with lead-anodes. By retarding the flow of this current through a second set of tanks, arsenic will be deposited with the rest of the copper. The slime deposited in such a purifying-vat showed Cu, 57.1 per cent.;  $\text{H}_2\text{SO}_4$ , 11.8 per cent.; Ag, 0.3 per cent.; As, 7.2 per cent.; Sb, 1.8 per cent.;  $\text{H}_2\text{O}$ , 18.4 per cent.; difference: traces of  $\text{PbSO}_4$  and traces of Fe.

While the difference in potential between electrodes with soluble anodes is 0.6 (Great Falls) and 0.3 (Anaconda) volts, with insoluble anodes it rises to 2.5 and 2.0 volts. When iron is to be removed, the solution, freed from copper and arsenic, is concentrated by heating to one-third of its volume and then cooled, when most of the iron will crystallize out in the form of ferrous sulphate. At Great Falls the cathodes are renewed every second day. They weigh 55 lb. and contain Cu, 99.95 per cent.; As, 0.0012 per cent.; Sb, 0.0033 per cent.; Ag, 1 oz. Although this method of working uses up a large number of starting-sheets, it is found to be advantageous, as with the large current of 40 amperes there is a loss in efficiency on account of short-circuiting if the cathodes are allowed to grow heavier. Thus with 2-day cathodes there is an ampere efficiency of 91 per cent., while with 4-day cathodes this falls off to 85 per cent. The cathodes, taken out 4 at a time by means of an overhead traveling-pulley, are transferred to lead-lined cars and replaced by starting-sheets. The 20 cathodes in a car are then raised together by means of an electric crane,

dipped into water, drained, dipped into milk of lime and allowed to dry. The coat of lime protects the plates in melting-down from the sulphurous gases in the products of combustion of the refining-furnace.

At Anaconda, cathodes are removed only when the anodes are completely corroded. After 2 days depositing they are, however, taken out and straightened. A tank receives all its anodes at one time, and in the same way all the cathodes are taken out by one operation.

The anode-mud made at Great Falls is removed by a steam-injector pump and rubber-hose from the electrolyzing-vat into a settling-tank covered with a 40-mesh screen; it is then drawn off into an acid-egg, forced through a filter-press, steam-dried (when it still retains 2 per cent. water), broken up, sampled, sacked and sold.

At Anaconda, the anode-mud is discharged through an opening in the bottom of the electrolyzing-vat, after the solution has been drawn off from the side, into a V-shaped trough leading into a tank; from this it is drawn into an acid-egg and forced into a tank in the refining-department, drawn onto filter-cloths, washed, dried, sampled, sacked and sold. Until recently the mud was treated at the works. It was melted down in a reverberatory-furnace, refined by the addition of niter, cast into small ingots and parted with sulphuric acid, the silver being precipitated by means of copper.

The anode-scrap at Great Falls is remelted in the converter, while at Anaconda it goes back into the anode-furnace.

The leading facts of the 2 refining-plants are brought together in Table XII., on page 314.

### *Refining of Cathodes.*

The reverberatory-furnaces used for refining converter- and cathode-metal have retained the pear-shaped form of the original Welsh furnaces. They have charging-doors on both sides, and the usual working- and ladling-doors at one end; and there are air-flues in the bridge leading onto the hearth.

The following are the leading details of a standard furnace: Length of hearth, 22 ft. 6 in.; width at bridge, 7 ft. 1.5 in.; at middle, 14 ft. 8 in.; at flue, 1 ft. 8 in.; hearth-area, 225 sq. ft.; thickness of hearth at bridge, 2 ft. 7 in.; at flue, 2 ft.; depth of metal, from 12 to 13 in.; length of grate, 8 ft. 6 in.; width,

TABLE XII.—*Details of Electrolytic Refineries.*

	Great Falls.	Anaconda.
Number of tanks.....	312	1,400
Length " ".....	9 ft. 9 in.	8 ft. 2 in.
Width " ".....	2 ft. 4 in.	4 ft. 5 in.
Depth " ".....	3 ft. 9 in.	4 ft. 6 in.
Number of tanks per set—i. e., number of of tanks using same solution.....	104	200
Number of anodes per tank.....	20	38
" " cathodes per tank.....	20	80
" " amperes per sq. ft. of cathode area.....	40	10
Power—		
Number of generators.....	2	7
" " circuits.....	1	7
Amperes generated per circuit.....	9,000	4,000
Volts per circuit.....	200	60
Character of material worked.....	{ Converter- anodes }	{ Casting-fur- nace anodes }
Kind of moulds used.....	Copper	Iron
Casting-machines, pattern.....	Walker	Anaconda
Converter-anodes, per cent. Cu.....	99.1	.....
" " " " As. and Sb.....	0.07	.....
" " " oz. Ag.....	56.00	.....
" " " oz. Au.....	0.20	.....
" " " per cent. scrap.....	8.00	.....
" " " mud, per cent. Cu.....	41.	.....
" " " " oz. Ag.....	7,000	.....
" " " " oz. Au.....	18	.....
Refined-anodes, per cent. Cu.....	99.27	99.25
" " " " As. and Sb.....	0.07	0.10
" " " oz. Ag.....	61.14	90.00
" " " oz. Au.....	0.22	0.50
" " " per cent. scrap.....	5.5	7.00
" " " mud, per cent. Cu.....	18.00	10.00
" " " " oz. Ag.....	15,000	18,000
" " " " oz. Au.....	38.00	100.00
Size of anodes—		
Length.....	35 in.	32 $\frac{7}{8}$ in.
Width.....	24 in.	24 $\frac{1}{2}$ in.
Thickness.....	3 in., 2 $\frac{1}{2}$ in.	1 $\frac{1}{2}$ in., 1 in.
Weight in pounds, converter.....	500	.....
" " " refined.....	632	230
Number of days corroding, refined-anode.....	.....	37
" " " " converter-anode.....	18	.....
Size of cathodes—		
Length.....	36 $\frac{1}{2}$ in.	33 in.
Width.....	26 in.	11 in.
Weight of starting-sheets, lb.....	2 $\frac{1}{2}$	$\frac{3}{4}$ , 1
Weight of cathodes, lb.....	55	100
Number of cathodes on one bar.....	1	4
Electrolyte—		
Sulphuric acid per liter, grammes.....	170	150
Copper per liter, grammes.....	42	40
Temperature at head tank, C°.....	64	50
Circulation per minute, gals.....	6	3
Difference in potential—		
Soluble anodes, per tank.....	.6	.3
Insoluble " ".....	2.5	2.
Composition of cathodes, per cent. Cu.....	99.95	99.96
" " " " " As.....	.0012	.0009
" " " " " Sb.....	.0033	.0023
" " " " oz. Ag.....	1.0	0.25

7 ft. 1.5 in.; grate-depth below bridge-top, 2 ft. 6 in.; ratio of grate to hearth-area, 1:3.7; height of roof above bridge, 1 ft. 10 in.; above hearth at bridge, 2 ft. 3 in.; at flue, 1 ft. 10 in.; rise of arch, 1 in. per ft.; size of flue at vulcatory, 3 ft. 9 in. by 19 in.; of flue leading to chimney, 24 by 19 in.; size of chimney, 32 by 32 in.; height, 80 ft.; charge: liquid, 125,000 lb.; cathodes, 75,000 lb.; analysis of bituminous coal: ash, 18.8 per cent.; fixed carbon, 54.6 per cent.; proportion of charge to coal by weight, 3:1; tons of charge to number of poles, 4:1; tons of charge to bushels of charcoal, 1:1.

The method of working differs little from that in common use, excepting that flapping, or rabbling, is replaced by blowing with compressed air (see *ante*, p. 309); the metal bath, in poling, is covered with charcoal only toward the end of the operation, a few old poles, however, floating on the metal; of the two poles inserted through the working-door, one is depressed near the door, the other toward the middle of the furnace; ingots, cakes and wire-bars are cast by means of suspended ladles, 14 in. and 19 in. in diam. All moulds are of copper. Cake- and wire-bar moulds are painted with a mixture of lamp-black and benzine; they are warmed at first by pouring in some copper. Ingot-moulds are smoked before the first round by means of burning rosin held in a ladle. The tests applied to the copper are those of the Bell Telephone Co. for No. 12 wire: Conductivity, 96 to 97 per cent.; tensile strength, 65,600 lb.; torsion, 40 twists in 6 in. between the clamps with 1 per cent. elongation. Three test-bars are made from each charge, one bar being made when dipping begins, one when the charge is half out, and the third at the conclusion of the ladling. The test-bars are 8 in. long, and have the full cross-section of a regular wire-bar. From each of these bars a section 6 in. long by 1 in. square is sawed; this section is rolled into a rod and afterward drawn cold to a No. 12 wire, Brown & Sharpe gauge, on which all tests are made. Conductivity-tests are made with the Willyoung conductivity-bridge.\*

In conclusion, I wish to express my thanks to the management and officers of the different companies for the courtesies shown me, and for the privilege of presenting to the Institute that part of my notes which is of general metallurgical interest.

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\* M. E. Leeds & Co., Philadelphia, Pa.

## APPENDIX.

TABLE XIII.—*The Efficiency of Some Rocky Mountain Coals.\**

	Summary of Chemical Tests.						Summary of Efficiencies.						
	Volatile Matter.	Fixed Carbon.	Total.	H <sub>2</sub> O.	Ash.	Calories.	B. T. U.	Price per Ton.	Evap. from and at 212° F.	Cost to Evap. 1000 lb. Water.	Relative Value.	B. T. U. of Coal.	Boiler Efficiency.
Trail Creek (Mt. House) nut.....	Per Ct. 25.92	Per Ct. 48.85	Per Ct. 74.77	Per Ct. 9.04	Per Ct. 16.19	7594	13,561	\$2.85	4.65	\$0.306	\$2.68	10,141	44.27
“ “ lump.....	25.34	52.62	77.96	8.75	13.29	7734	13,921	4.15	5.70	0.360	3.28	10,863	50.00
Red Lodge lump.....	26.24	52.39	78.63	9.23	12.14	7245	13,041	4.50	6.50	0.345	3.74	10,255	61.00
Bridger lump.....	26.55	54.00	80.55	7.67	11.78	7465	13,437	4.50	6.10	0.371	3.51	10,824	54.30
Sheridan, Wyoming, lump.....	26.78	46.40	73.18	21.20	5.62	7006	12,611	5.50	5.01	0.547	2.88	9,929	52.40
Red Lodge washed nut.....	26.15	53.52	79.67	9.05	11.28	7205	12,970	3.75	5.40	0.346	3.11	10,332	50.60
Chestnut washed mine-run.....	23.10	50.18	73.28	8.64	18.08	8407	15,133	3.05	5.30	0.288	3.05	11,090	46.50
Gebo lump.....	21.46	47.40	68.86	6.42	24.72	8241	14,834	4.00	5.75	0.346	3.31	10,215	54.40
Bridger nut.....	30.00	50.64	80.64	3.95	15.41	7195	12,952	3.75	4.86	0.386	2.80	10,444	44.80
Trail Creek (Kounitz's) lump.....	26.24	47.65	73.89	12.41	13.70	7167	12,901	4.50	5.44	0.413	3.13	9,533	55.00
Belt washed nut†.....	22.65	55.00	77.65	4.23	18.12	7445	13,402	4.50	5.60	0.400	3.22	10,406	52.00
Galt (Canada) lump†.....	24.28	57.45	81.73	8.00	10.27	7421	13,359	6.50	6.05	0.535	3.49	10,919	53.10
Rock Springs (Wyo.) lump†.....	27.90	61.31	89.21	5.64	5.15	7686	13,836	6.75	7.55	0.447	4.35	12,343	59.00
Mountain Side mine-run.....	26.84	49.91	76.75	2.71	20.54	7918	14,252	3.40	5.78	0.294	3.32	10,938	51.00

\* Taken from a paper read before Montana Society of Engineers by Prof. W. H. Williams, State Agricultural College, Bozeman, Mont., through *The Western Mining World*, November 15, 1902.

† Price in Butte. Relative value compared to chestnut washed mine-run.

**Note Concerning an Old Instrument for Finding Distances,  
Exhibiting the Oldest Known Form of the Transit-  
Theodolite Principle.**

BY H. D. HOSKOLD, BUENOS AIRES.

(New York Meeting, October, 1903.)

DURING the last few years, various persons have been put forward as originators of some mechanical device for the purpose of finding distances without the use of a chain or other linear measuring-instrument, and among these, the Englishman, Gascoigne, 1639, is prominent as the first in England; while James Watt, 1771, and William Green, 1778, are also included in the list of inventors.

It is possible that this useful and important problem occupied the attention of others prior to the date cited, and, consequently, it is difficult to decide what nation first gave birth to the inventor of an instrument for the object under consideration.

A very important work appeared, not long since, in Paris, devoted to a history and discussion of some of the older and newest forms of surveying-instruments,\* and at page 79 of that work a curious old instrument is exhibited, called *Trigonometre*, *i. e.*, an instrument for finding distances and solving trigonometrical problems in surveying. A copy of the diagram of the instrument exhibited in the book referred to is represented in Fig. 1.

Having had an occasion to read the French work referred to, I communicated with the author of it, and he was good enough to reply that he found a diagram of this old instrument in a book published in 1597, by Danfrie, existing in the library of the Academy of Sciences, in Paris. Col. Laus-

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\* *Recherches sur les Instruments, les Méthodes et le Dessain Topographiques*, Col. A. Laussedat, Paris, 1898.



sedat further said, that he accidentally found a second copy of Danfrie's book in a second-hand book-shop, and also that the instrument of Danfrie is described in an old Dutch book, the title of which he had forgotten.

In arranging my library recently, I came upon some old Italian literature, and on examination I discovered in one of the books a very curious old instrument constructed for the object of finding distances.\* The diagram illustrating this instrument is found on page 9 of the book in question, and is represented by Fig. 2 of this paper.

It appears from the date of publication that this instrument was devised prior to 1564. The old Italian book referred to is an authoritative document of considerable scientific interest and value, and enables us to conclude, with a high degree of certainty, that this Italian instrument, called *holometro*, although similar to the French one of Danfrie of 1597, is still older, more practical and perfect, and also that either it, or some older one, must have formed the original model from which, in all probability, that of Danfrie was derived.

We find that Fullone published his book in French prior to 1564, and it is evident that he included in it the instrument, *holometro*. From the introduction of the old Italian work, it appears that a manuscript copy of the French book of Fullone was taken to Venice and reproduced there by Giordano Ziletti, in a copy of which I found the diagram of the *holometro*. The word *ritrovato*, occurring on the title-page, is evidence that Fullone was the inventor or discoverer of the instrument, and, considering that he bases its use upon Euclid's grand proposition of similar proportional triangles, Fullone must have possessed considerable ability as a mathematician. There is no evidence to show that he knew of the existence of any similar instrument previous to his invention, and this is strengthened by the fact that Ziletti, in 1564, reproduced Fullone's work, as previously noted.

No doubt, however, the earlier-constructed astrolabes may

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\* *Descrittione ed Uso Dell' Holometro per saper misurare tutte le cosse, che si possono vedere coll'occhi così in lunghezza e larghezza; come en altezza e profundita. Ritrovato per Abel Fullone, valetto di camera del Re di Francia. En Venetia, appresso Giordano Ziletti, al Segno della Stella, 1564.*

have aided Fullone in his invention; but such instruments, as far as I am aware, were in general freely suspended or held in the hand when in use, and not attached to a rigid perpendicular leg, or stand, much prior to the time of Rojas, 1550.

Fig. 2 is a reduced copy of the *holometro* as it is found in Ziletti's book. It consists of various distinct parts, and, for that period, must have been considered a very important, although somewhat complicated, instrument.

The table, or base-plan, of the instrument has the form of a geometrical square—size unknown—at the centre of which was placed a small compass-box containing a magnetic-needle. The inner circle is divided into 32 equal parts, each being equal to  $11^{\circ} 15'$ , similar to the present mariner's compass. The next circle is divided into 80 equal parts of  $4^{\circ} 30' 00''$  each; and a third circle is divided by radial lines from the center into 16 parts of  $22^{\circ} 30'$  each. In another larger figure occurring in the same book, representing this part of the instrument, a still larger circle is represented, inscribed to fit the inside limits of the horizontal table, or geometrical square, and divided into *four* parts, representing the cardinal points.

The lower, or nearest, edge, or base of the square of the instrument, has a long slip of iron inlaid and divided into 10 principal parts, and on a parallel line; each of these is divided into two parts and figured from 10 to 100. A third inner parallel-line scale exhibits each of the latter divided into 5 equal parts.

At the right- and left-hand extremities of this divided base, or scale, a long, metal, radially-divided rule, or bar, *regla*, is mounted upon a pin or axis, and each of these radial bars is divided into 190 equal parts of the same scale as the base previously noted. At the one-hundredth division on each of the bars a hinged joint was formed, and when the divided bars or rules were made to cross each other at these points, the two divided bars and base formed an equilateral triangle. The joints, also, permitted the divided bars to be folded and to fit the form of the table for facility in transport and packing in a case.

Upon the axis of the left-hand radial bar a metal quadrant was fixed, and divided into 24 equal parts of  $3^{\circ} 45'$  each,

and this arc, axis and divided rule could be slid along a groove towards the right-hand rule, with a view of shortening the base of 100 equal parts, when the distance to be determined was great, and the bars would not cross one another with the standard base of 100 parts.

Fig. 3, *Il Rettore*, exhibits a divided, vertical metal circle mounted upon an axis and fixed to the upper part of a perpendicular leg, or stand. The circle was divided into four quadrants, each of which was subdivided into 24 equal parts. The figure, however, shows only the upper half to be divided; but in the original description it is stated that the entire circle was divided. A stout bar with plain sights was mounted and revolved upon an axis which passed through the center of the circle, and was made to revolve vertically around the entire circle, thus measuring upon it angles of elevation and depression.

Two such vertical circles were used with the *holometro*, one being mounted on each of the long divided rules. A dovetailed recess was formed upon each of the divided rules between the 13th and 28th divisions, or near to the axis of revolution, and the base-part of a perpendicular leg, or stand, carrying the vertical circle, was slipped into the dove-tailed recess on the left, and one also into the right-hand bar or divided rule; thus the legs, or stands, with the vertical circles were easily attached and detached. A plumb-line was suspended from the center of each of the vertical circles, for leveling the entire instrument. The whole affair was mounted upon a stand with a ball-joint at its top, and a clamp-screw to make all fast.

Ziletti refers to a cut (Fig. 4) in order to explain the use of the *holometro*. The instrument was set up at the first station, to the right, as indicated by the letter A. The right-hand divided rule was then turned on its horizontal axis and directed by means of the plain sights of the vertical circle to the second station, B, on the left, where the surveyor is afterwards stationed. The line of sight passing to this point is made to coincide with the line of the divided base of the instrument.

Then, without moving the table or base, the divided rule was turned on its axis and a sight taken to the tower at C.

The right-hand divided rule was then left in this position, and the direction pointed out by the magnetic-needle was observed.

The instrument was then erected at the second station, B, and the measured distance between the two stations was taken at 100 steps, or units of measure. The table was then moved horizontally upon its ball-joint until the magnetic-needle pointed out the same bearing as that observed at the first station, and the left-hand divided rule was directed to the tower at C, by sighting through the plain revolving-sights of the vertical circle. The point where the two rules crossed each other gave the distance from A to C, and the left-hand divided rule the distance from B to C. Ziletti says :

“If the rule to the right is crossed at the division of 190, the surveyor determines that from A to C there are 190 steps. In the same manner, the division intersected or cut on the left-hand rule by the crossing of the first indicates the distance from the second station, or B to C.”

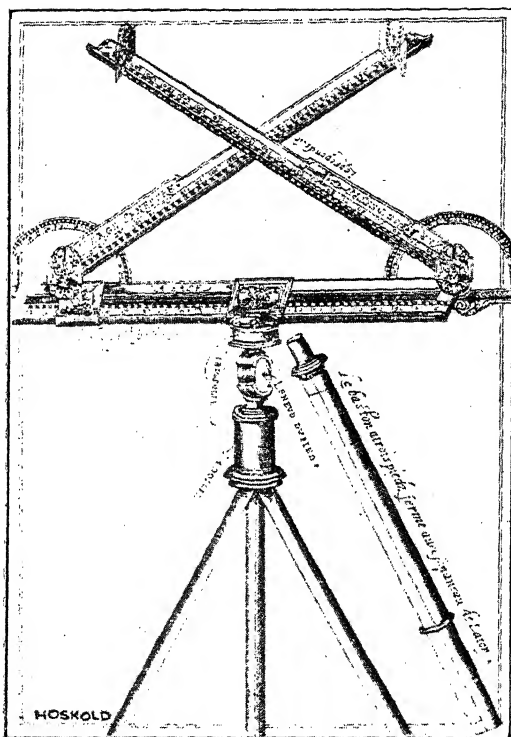
When the distance is too great, and after sighting the two rules do not cross one another, the left-hand one, with its quadrant, was slipped along the divided horizontal metal base, or towards the 100th division, so shortening the base until the rules crossed each other after the two observations were made; and from the divisions on the rules and those on the shortened base the distance was determined by the rule of proportion.

The *holometro* was also applied in determining the distance and height of a tower. It is said in the old book that the instrument may be employed for mine-surveying.

In the preceding description we have evidence that the plan of mounting a graduated vertical circle, with plain revolving-sights upon a horizontal axis, and attached to a perpendicular stand, or leg, for measuring angles of elevation or depression, was known and applied by Fullone prior to 1564, or more than 300 years since.

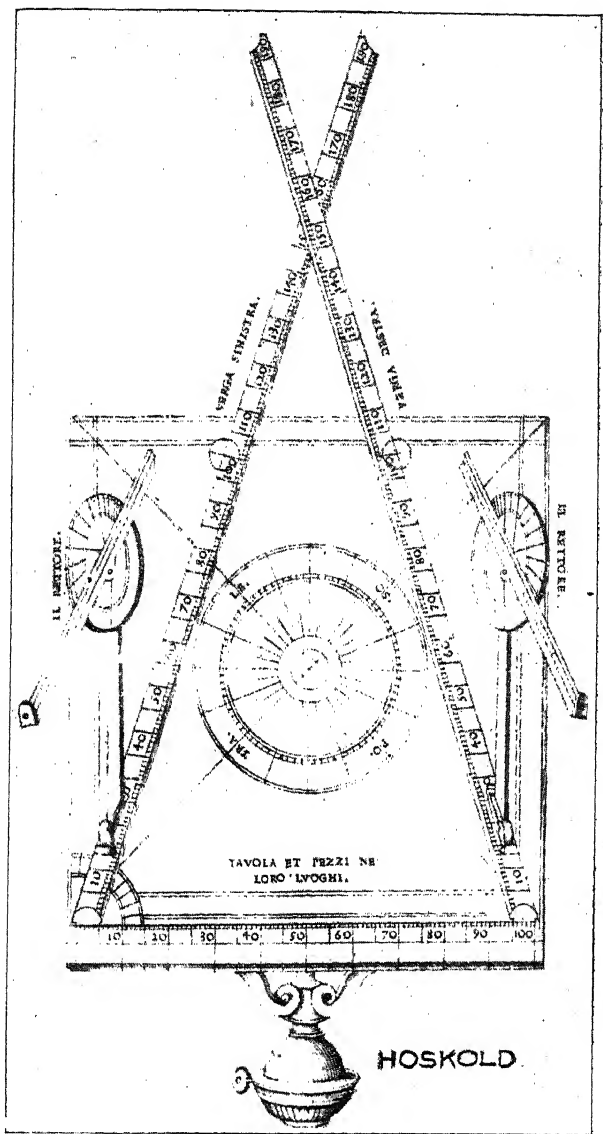
This principle then having been introduced, the transition was slight from such an original model, if it really were such, to the plan afterwards generally adopted for supporting the horizontal axis of a vertical circle at its two extremities upon Y legs or standards, such as have been employed in the con-

FIG. 1.



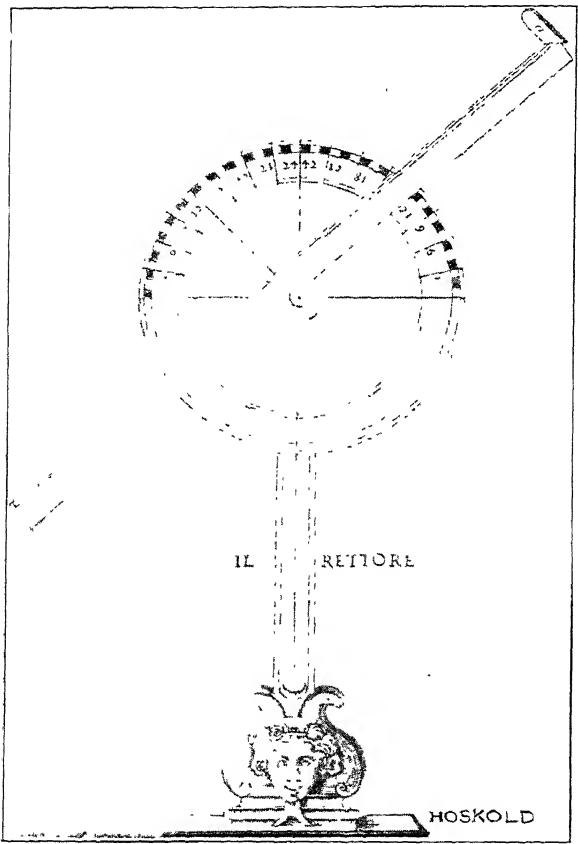
Trigonometre of Danfrie (1597).

FIG. 2.



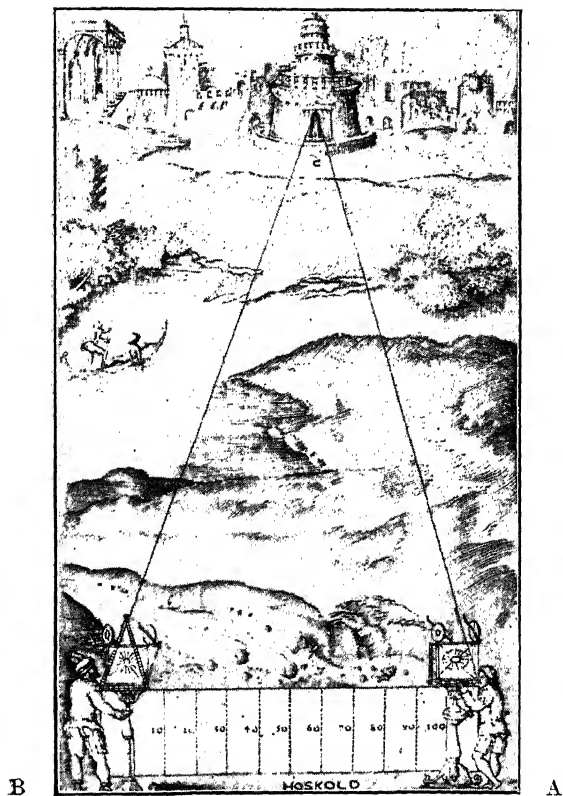
Holometro (prior to 1564).

FIG. 3.



The Rettore.

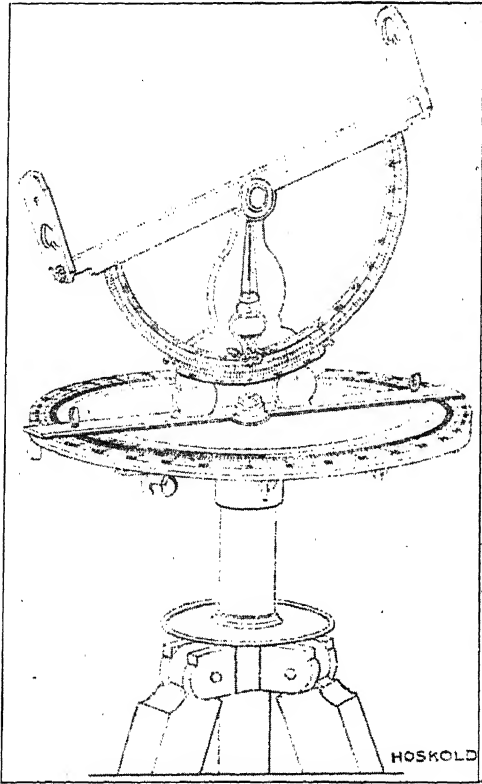
FIG. 4.



Method of Using the Holometro.

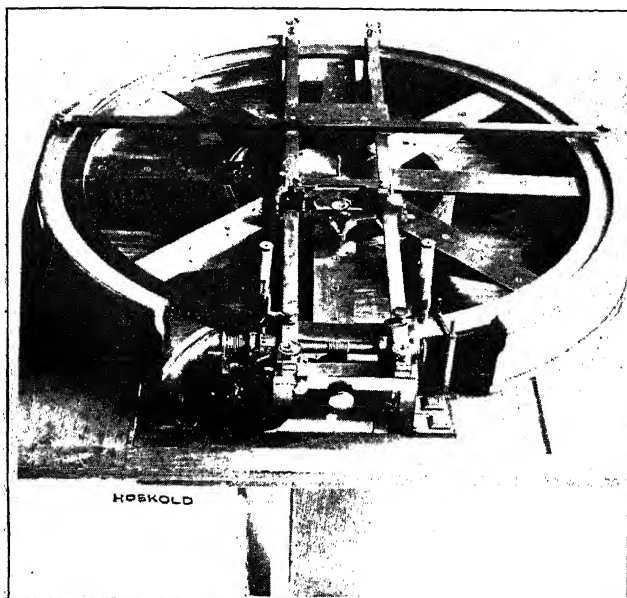


FIG. 5.



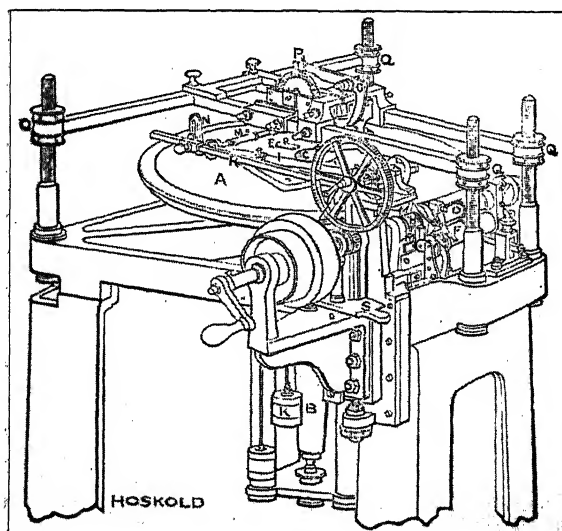
Old English Theodolite (about 1632).

FIG. 6.



Ramsden's Dividing-Engine.

FIG. 7.



Dividing-Engine (Troughton and Simms).

struction of alt-azimuth circles and modern portable transit-theodolites. The change was induced by reason of the extra weight of the instruments and the great amount of precision required by their use.

We have it on record that Römer, 1700; Halley, 1719; Ramsden, 1789; Troughton, 1789–1792, and Cary, 1802, are included in the list of those who constructed, or caused to be constructed, instruments upon the transit-principle.

Referring to the originators or the first constructors of portable surveying transit-theodolites, Mr. Casella, the well-known mathematical instrument-maker of London, communicated the fact that, “in 1776, Ramsden made portable transits, and, no doubt, one could be found if trouble were taken to do so.”

Col. Laussedat, the celebrated scientific writer of Paris, previously referred to, has illustrated at page 105 of his work *Recherches*, etc., what he calls an “ancient English theodolite,” but he does not give the name of the maker or the date of construction. It is represented in Fig. 5.

Considering that much interest and importance is attached to this instrument, it has been reproduced because it seems to fill a gap, and illustrates the progress made in the construction of instruments from the time of Digges, 1571–1591, to that of Sisson, who constructed theodolites of a peculiar form prior to 1727. It will be observed that Fig. 5 shows a vernier attached to the vertical semicircle, which circumstance is sufficient to place the date of its construction not earlier than 1631; and it is highly probable that it was later than that year.

Our friend Col. Laussedat states another important fact, and that is, that he found the *graphomètre* first described, and illustrated, in a work of the date of 1597, by Philip Danfrie. This instrument had two sets of sights, one pair being fixed to sight the line of the diameter of the horizontal semicircle, and the other pair movable around the central axis. This instrument seems to have been more popular in France than the *trigonomètre*.

*Ramsden*.—On former occasions, reference has been made to this ingenious and celebrated practical mathematical instrument-maker, and some confusion as to dates has crept into the literature treating upon the subject.

At the age of 23, Ramsden was apprenticed to a mechanical instrument-maker in London; but who that maker was is not known. Ramsden was most successful in his new pursuit, and was much sought after by all the other makers in London at that time. He opened a mathematical instrument shop in the Haymarket, and afterwards removed to Piccadilly, where he remained until his death.\*

Various important instruments, including his great 36-in. theodolite, were invented and constructed by him, and many other different classes of instruments were improved; but his greatest achievement, due to the extraordinary skill and perseverance which he possessed, was in the invention and construction of his original and memorable circular dividing-engine. This machine occupied his attention for several years, and was not brought to a satisfactory condition until after many failures. His first dividing-engine was produced about 1768, but was not perfected until 1773-4. An account of it was published in 1777.

Ramsden had several apprentices, and among others William Cary, Edward Pritchard and Thomas Jones. The first- and last-named were afterwards celebrated as instrument-makers.

The favorite apprentice of Ramsden was, however, Edward Pritchard, whom he called *Ned*, and to whom he left a dividing-engine at his death. Edward Pritchard died, leaving three sons, and the dividing-engine of Ramsden between them. One of the sons arranged with Simms for the purchase of the dividing-engine; and afterwards this son, Pritchard, sold the same engine to Dollond, the optician of St. Paul's churchyard, and father-in-law of Ramsden. Afterwards, Pritchard went to North America, and no more was heard about him. Some opticians in London, however, still entertained the idea that he took the engine, formerly sold to Dollond, to America with him; but the balance of evidence is against this idea. When Dollond sold his business, the dividing-engine which he had purchased from Pritchard was put up to tender and was bought by a member of the Parsons family, dividers and en-

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\* The *Penny Cyclopædia* states that from 1766 to 1784 Ramsden's shop was in the Haymarket; but in the latter year he removed to Piccadilly, where his business was carried on after his death.

gravers, and the machine, then purchased, is now in the possession of Mr. Walter Parsons, divider to the trade, London.

It appears that this machine is still in excellent working-condition. Mr. Parsons says that it is "as finely constructed a piece of mechanism as the world could produce;" and Casella, when referring to the same subject said, "Ramsden's dividing-engine is here, in London, and it is a beautiful piece of work, such, indeed, as one seldom sees."

Mr. Walter Parsons has in his possession another dividing-engine, which he says has always been considered to have been made by Ramsden. Mr. Parsons possesses four other dividing-engines, one of which was constructed by Allen, who gained for it in 1810 the gold medal of the Society of Arts, and also a premium of £100 for the best divided circle, which he graduated upon his engine. One of the engines possessed by this firm was made by the mathematical instrument-maker, Anthony Lear.

Mr. Parsons stated that John Troughton, the original founder of the present firm of Troughton & Simms, made the (dividing) engine which the original firm of Parsons used, and which was bought by the present member of the family. The original or first dividing-engine of Ramsden was, until recently, in the possession of the Royal Society of London, and it is said to have been presented to the government, and at one time to have been in the possession of the South Kensington Museum.

With his usual courtesy, Col. Laussedat informed me that one of the dividing-engines of Ramsden for dividing astronomical circles found its way into France, and was, according to an existing inventory, consigned to the establishment *Conservatoire National des Arts et Métiers*, in the year 1795; but the present director of that establishment does not seem to be able to determine in what manner the dividing-engine in question was originally acquired. It is highly probable that the French government, or some scientific society, may have ordered the engine direct from Ramsden.

In accordance with my instigation, Col. Laussedat made particular inquiries from the present director of the *Conservatoire*, referring to the question of dividing-engines, and in reply it has been communicated that there exists, in this establishment,

another similar machine constructed upon the same principle as that of Ramsden, by Samuel Rhéi or Rhée, and introduced into France by Andréossy. It was received in the *Conservatoire* in 1810, and, according to the opinion of the director of that establishment, must have been presented by the Academy of Sciences, Paris.

It is also declared that there is in the archives of the said establishment an unsigned document, containing the following notice :

“ In 1810, the government made a present to the *Conservatoire des Arts et Métiers* of a machine [dividing-engine] of the largest dimensions, adapted to the dividing of the circle in any number of divisions which may be required.

“ This machine was constructed by Samuel Rhéi upon the same principle as the one invented by Ramsden, which has become a treasure for the English navy. This important dividing-engine combines the last perfections possible, especially that made in the *regulator*, which serves to determine the number of divisions required to be engraved upon a circle, in a sure, prompt and precise manner. The proofs we have made [with this machine] have convinced us that it is possible by this process to divide instruments, *i.e.*, circles, with as much promptitude and regularity as the author could obtain by his machine, and must be regarded as the most perfect. In fact, this machine, considered under the double advantages of composition and execution, presents to the pupil who might have to deal with the construction of machines, the solution of problems of the most difficult kind in mechanical practice.”

In the same establishment also exists a wooden model of the dividing-engine of Ramsden.

The notice of the *Conservatoire Nationale* is exceedingly interesting, because it proves that the dividing-engine of Ramsden, referred to by Col. Laussedat, was the first machine of that class possessed by the French ; that it led the way to the construction of others, and also introduced the more accurate system of dividing circles in France by mechanism instead of by hand. The director of the *Conservatoire National des Arts et Métiers*, Paris, kindly presented to me a photograph of the Ramsden dividing-engine conserved in that establishment, a copy of which is exhibited in Fig. 6.

The technical foreman of Ramsden was named Bergh, and he is reported to have made, or finished, a dividing-engine which Ramsden was unable to complete, before his death in 1800. The business which Ramsden had established at Piccadilly, near to St. James's church, was afterwards carried on

under the style of Bergh, late Ramsden. Bergh was succeeded by Worthington & Allen, and at their death the business ceased.

Two of the principal apprentices of Ramsden—William Cary and Thomas Jones—turned out to be excellent mathematical instrument-makers. The former is still represented under the same name and by a descendant of Cary; and although Thomas Jones had great reputation and business under the patronage of the Admiralty, after his death in the hands of his son the business collapsed.

The instruments constructed by Thomas Jones were divided by the Parsons family.

Mr. Stanley says that he has an excellent dividing-engine constructed by himself, and he further adds that it is generally understood that the best dividing-engine in London is the one constructed and used by Troughton and Simms, a description of which, with a diagram, appeared in the last edition of the *Encyclopædia Britannica* (Fig. 7).

Troughton & Simms' great 36-inch Indian geodetic theodolite was subdivided by micrometers to read to 0.01 second, the dividing having been done by this engine; and, after many trials, the late Col. Strange, Inspector-General of Instruments for the Geodetic Survey of India, could not discover a single second of difference between each successive pair of divisions.

Mr. Walter Parsons says that he possesses a very high-class dividing-engine constructed by himself.

Since the time of Ramsden, the mechanical art has acquired great perfection, offering more facility and accuracy in use, which, together with the advanced technical condition of the higher-class workmen, renders it possible for every notable mathematical instrument-maker to possess a dividing-engine. The firm of Buff & Berger, instrument-makers, of Boston, possess two dividing-engines, one with 2-ft. circles, made by Würdemann and tested by Siegmüller, and the other with 3-ft. circles, made by J. H. Temple, of Boston.

If Dollond possessed two dividing-engines, which is doubtful, then it may be true that Pritchard took one of them to North America; and in that case the engine must still be possessed by some maker or museum in the States, and, consequently,

one would suppose that it would not be a difficult matter to trace it, as, also, the beginning of the system of dividing-instruments now followed.

A pamphlet was published in Washington, the date of which I have forgotten, upon the dividing-engine of Ramsden, a copy of which was offered for sale in Germany; and it is believed that it fell into the hands of Mr. Dunbar D. Scott. Probably, therefore, that gentleman, or the author of the pamphlet, or some other older firms of instrument-makers in the States, may be able to give some information relative to the time when Mr. Pritchard arrived in America, where he was employed as a divider or in some other capacity, and whether or not the story is true that he took with him the dividing-engine which, according to the evidence adduced, he had previously sold to Dollond, or any other.

I desire to express my great obligation to the following gentlemen, and to offer them my thanks for the generosity they exhibited in searching for valuable information, a good portion of which has been incorporated in this notice:

The Director-General of the British Ordnance Survey; the Astronomer Royal, Greenwich; Col. Laussedat, Paris; the Director of the Conservatoire National des Arts et Métiers, Paris; and the long-established firms of mathematical instrument-makers, Messrs. Troughton & Simms, Messrs. Elliot Bros., Messrs. Cary & Co., Mr. Wm. F. Stanley, Messrs. Casella, Mr. J. H. Stewart, Messrs. John Archbutt & Sons, London, and Messrs. John Davis & Son, Derby, besides other personal friends.



## The Treadwell Group of Mines, Douglas Island, Alaska.

BY ROBERT A. KINZIE, ASS'T SUPT., TREADWELL CITY, ALASKA.

(New York Meeting, October, 1903.)

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## INTRODUCTION.

The Treadwell Group comprises five mines, four of which are situated on the northeast side of Douglas island, directly opposite the town of Juneau, and one about four miles back of Juneau. These mines are owned by four separate companies, but are all under the same general management. Those situated on Douglas island are: The Treadwell mine, owned by the Alaska Treadwell Gold Mining Co.; the Mexican mine, owned by the Alaska Mexican Gold Mining Co.; and the Ready Bullion and Seven-Hundred-Foot mines, owned by the Alaska United Gold Mining Co. The Juneau mine, owned by the

Alaska Juneau Gold Mining Co., is on the mainland. This property embodies a large number of claims and promises to be very valuable; but it has not advanced beyond the prospect-stage, and as it differs entirely from the island mines, both in geological features and methods of working, it will not be considered in the present paper.

The principal topographic features of the country surrounding the Treadwell group of mines are well shown in the half-tone illustrations, Figs. 1, 2 and 3. Fig. 1 is a view of the works of the Alaska Treadwell Gold Mining Co. on Douglas island; Fig. 2 is a view of the Treadwell open-pits or "glory-hole" looking eastward; and Fig. 3 is a view from the eastern end of the Treadwell open-pits looking toward the towns of Douglas and Juneau which are plainly seen in the distance. An ideal cross-section of the Treadwell mine, illustrating the method of removing the ore, is given in Fig. 4, and a general map of the mines of the Alaska Treadwell Gold Mining Co., the Alaska Mexican Gold Mining Co. and the Alaska United Gold Mining Co., all on Douglas island, is shown in Fig. 5.

The features that have made the Treadwell mine justly famous are the mining and milling, at a profit, of ore that does not yield an average of more than \$2 per ton.

*Climate.*—Contrary to the accepted idea, the climate is not in the least severe, the winters being much milder than in New York, while the summers are never hot. During the last two winters the thermometer did not reach zero. The precipitation, however, is great, the average annual rainfall for the last three years being about 98 in. As a result of the mild climate and abundant rainfall, the entire country is covered by a dense growth of trees, consisting principally of hemlock and spruce, which makes fairly good mine timber, but poor lumber.

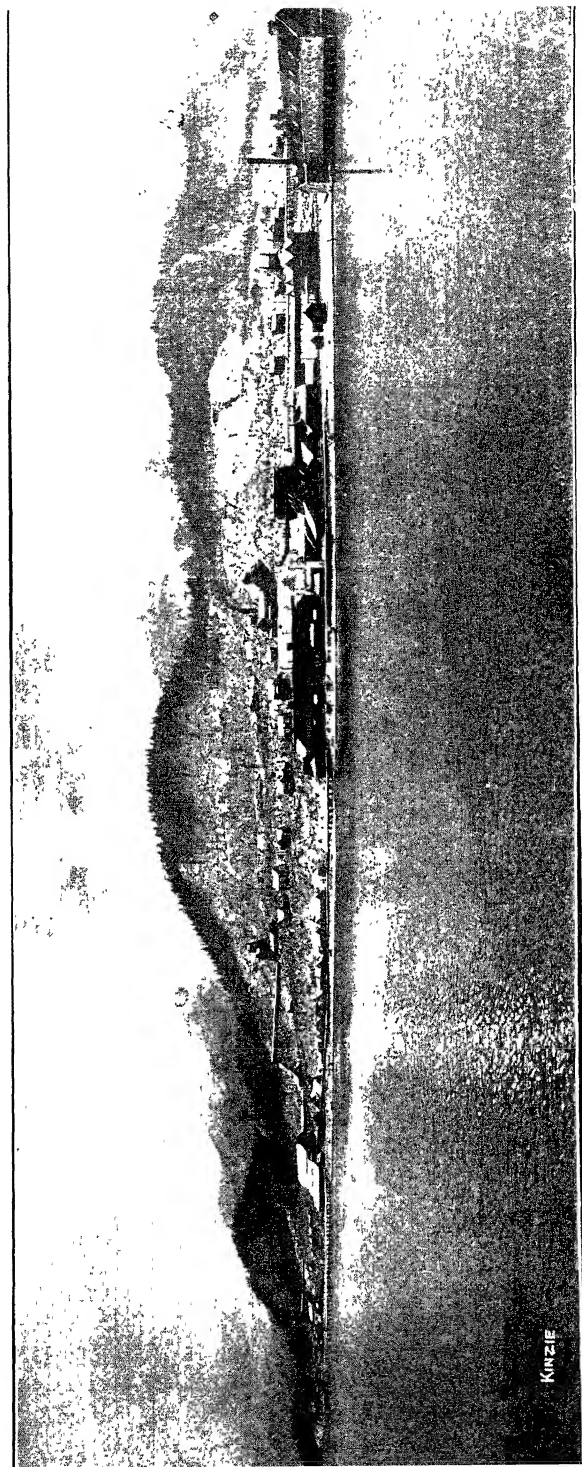
*History.*—Like every other gold-deposit in Alaska, the Treadwell mines were first worked as placer-diggings, and in this way was discovered the quartz in place, now known as the Treadwell deposit. From all accounts these surface-workings were quite rich, but were shallow and soon exhausted. It was in the summer of 1881 that gold was first discovered and placer-mining started. The placers in the Silver Bow basin, just across the channel, which had been discovered the previous year, proved exceedingly rich for a short period. Up to this

time there had been no attempt at systematic mining in Alaska. Soon after the discovery of the placer-deposits on Douglas island, the Paris claim, which covered the site of the large surface-pit, or "Glory Hole," of the Treadwell mine was bought from French Pete by John Treadwell for the sum of \$400.

Owing to the more than tropical luxuriance of the vegetation and the deposit of glacial mud which covers the surface to a depth of 20 ft., early exploration and development were extremely difficult. In 1881 the first mill, containing five stamps, was built and started crushing ore that came from a streak about 20 ft. wide, which runs along the foot-wall of the deposit. This ore is said to have milled between \$8 and \$10 per ton; at any rate, it was sufficiently high-grade to pay for the mill-and development-work and to leave a good surplus. It was early seen that the great bulk of the ore was extremely low-grade, and the fact was recognized that a large tonnage and, consequently, a large milling-capacity would be required to make the enterprise a success. With this idea in view a 120-stamp mill was erected and started crushing ore in June, 1885. The results of the first three years' run warranted an increase of 120 stamps, and the 240 stamps were put in operation in August, 1888. After a careful study of the results of running the 240 stamps, and of the ore-reserves in the mine, it was again decided to increase the milling-capacity by adding 300 stamps. Work was immediately started, and the 300 additional stamps were dropping in May, 1899, thus making a total of 540 stamps crushing ore from the Treadwell mine.

Soon after the opening of the Treadwell, the deposit was traced along its strike to the east, and the ore-body of the Mexican mine discovered. Unlike the Treadwell and Ready Bullion croppings, the surface-showing of the Mexican did not indicate the size of the ore-bodies that were afterwards opened up. As in the Treadwell, some fairly high-grade ore was found near the surface, but it was evidently a concentration, and did not represent the true or average value of the milling ore. It was not until 1894 that the mill of 60 stamps started crushing ore, but the result of the first year's milling was such that it was determined to increase the number of stamps to 120. This work was immediately started, and in 1896 the full mill was in operation.

FIG. 1.



VIEW OF DOUGLAS ISLAND, SHOWING THE WORKS OF THE ALASKA TREADWELL GOLD MINING CO.

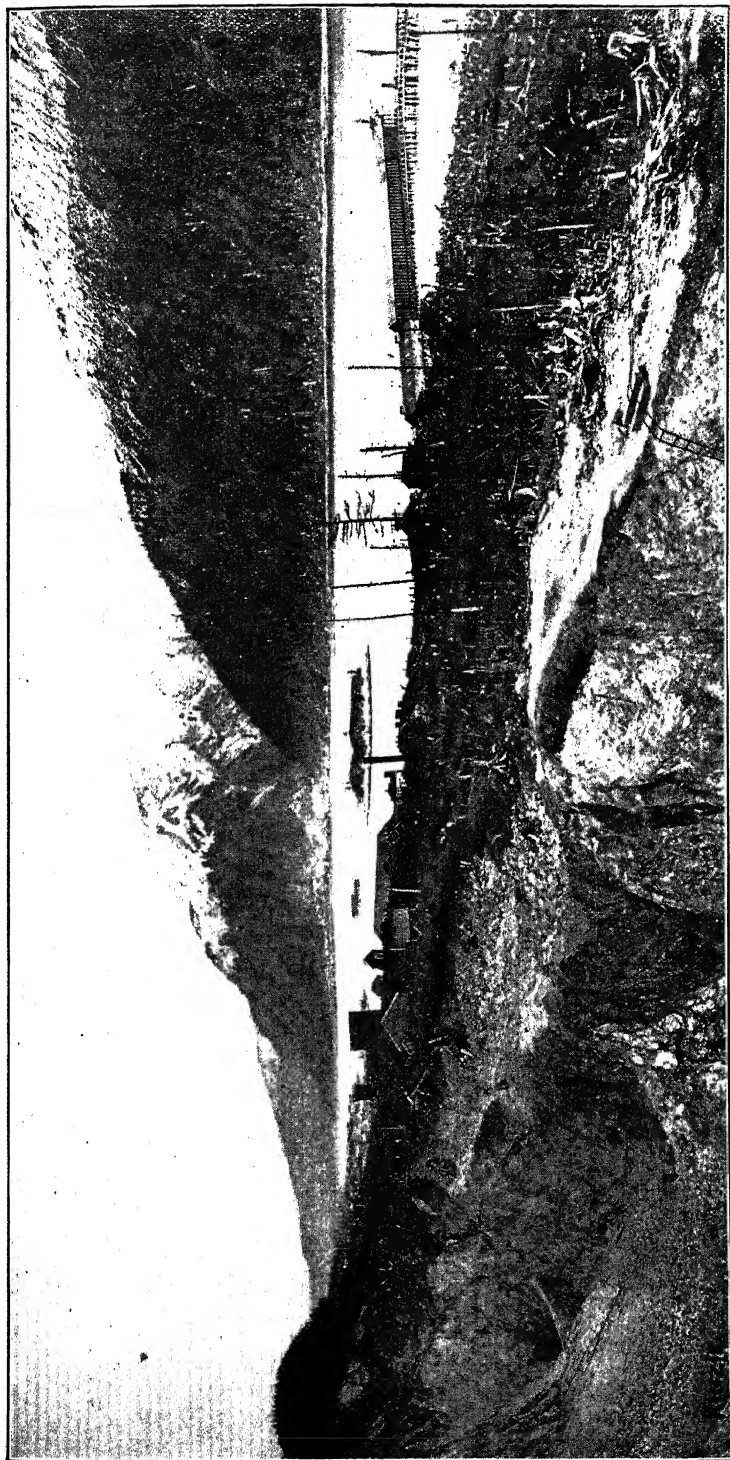
KIRZIE

FIG. 2.



TREADWELL OPEN-PITS OR "GLORY HOLE," LOOKING EASTWARD FROM PIT NO. 4.

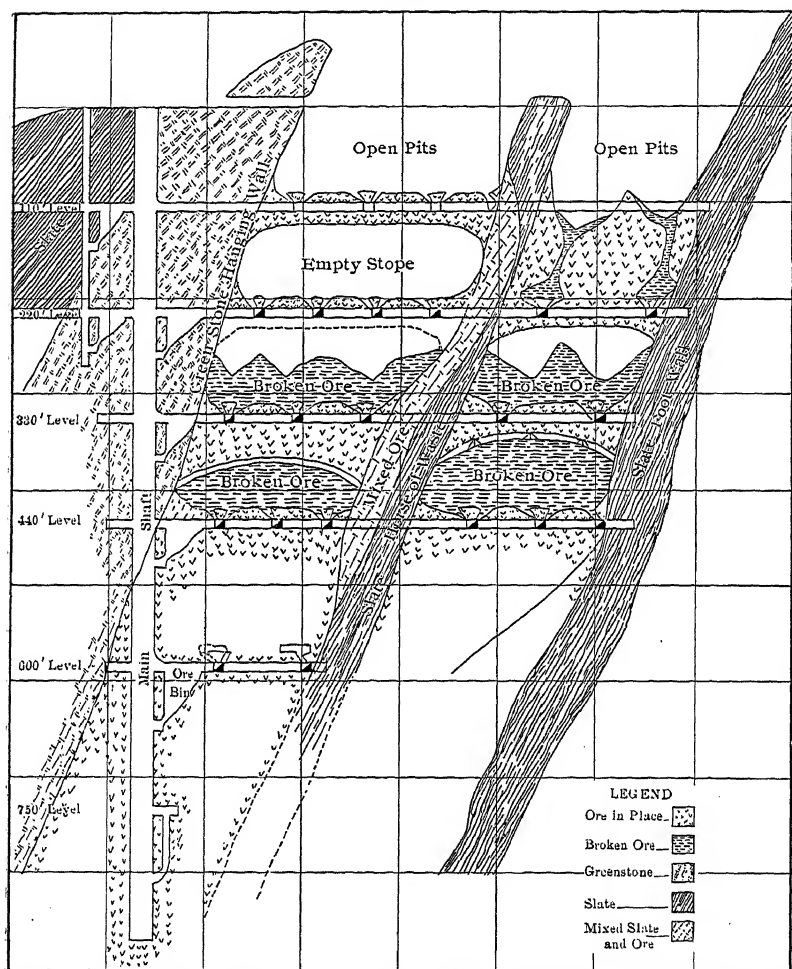
FIG. 3.



EASTERN END OF TREADWELL OPEN-PITS, SHOWING DOUGLAS AND JUNEAU IN THE DISTANCE.

After the discovery of the Treadwell and Mexican mines, there was a rush of prospectors into the district, and it was not long before rich placer-workings were found over what is now known as the Ready Bullion mine. For the most part, these

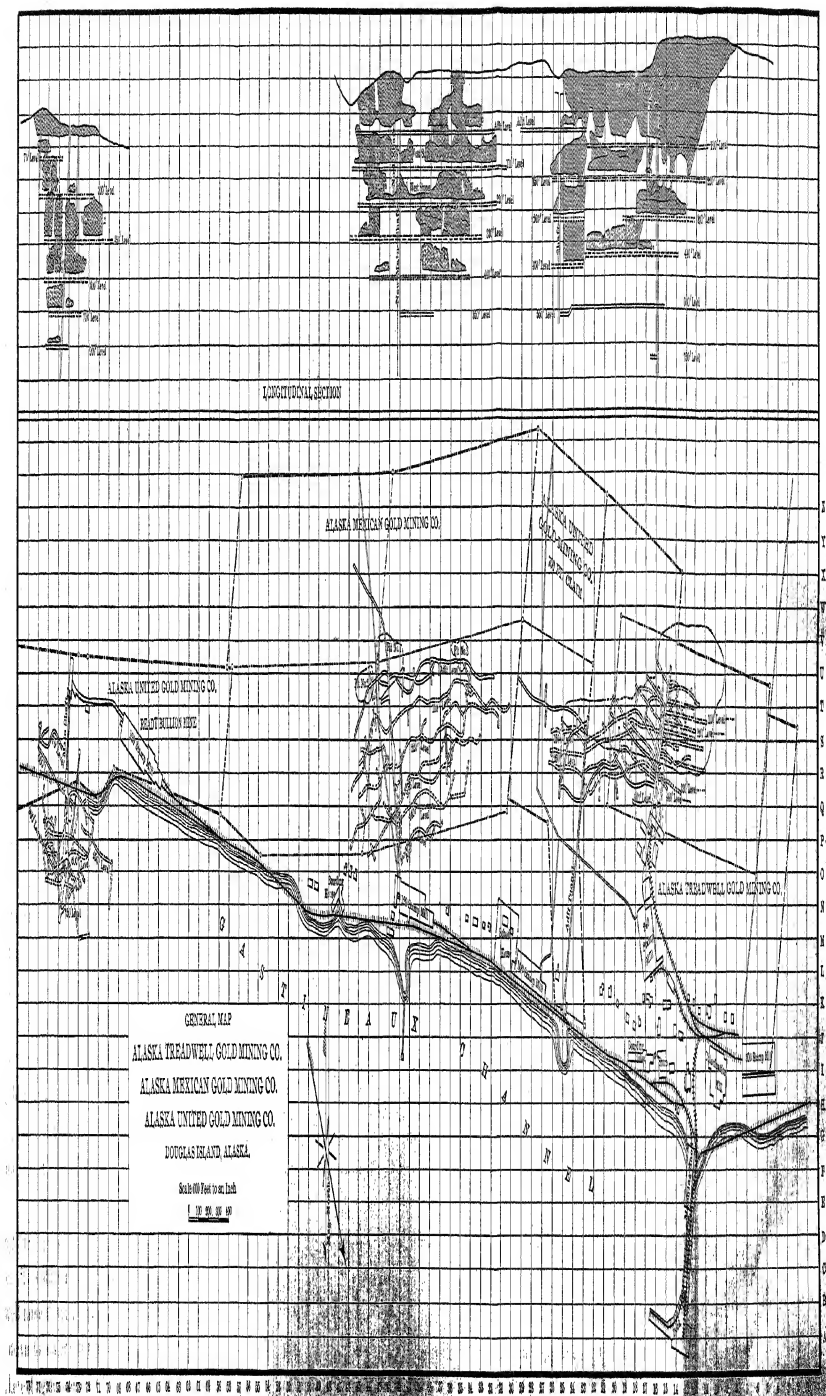
FIG. 4.



IDEAL CROSS-SECTION OF THE TREADWELL MINE, SHOWING THE MANNER OF WORKING THE DEPOSIT.

workings were on the beach below high-water mark. The gold was much coarser and the ground richer than in either the Treadwell or Mexican surface-workings.

Nothing was done here to open up the mine until 1894, when







it was acquired by the Alaska United Gold Mining Co., which company had also gained control of the extension or the end of the Treadwell ore-body, now known as the Seven-Hundred-Foot mine.

In 1898 a 120-stamp mill was erected and started crushing ore from the Ready Bullion, and in the following year the 100-stamp mill of the "700-Foot" was put in operation, thus making a total of 880 stamps dropping on the four properties.

*Geology.*—Before taking up the subject of the underground mining in detail, it would be well to give a brief description of the geological features that bear on the economic extraction of the ore. While the geological conditions are the same in the Treadwell, Mexican and Ready Bullion mines, there are conditions due to faulting, shape and the position of ore-bodies with regard to salt water, that have a great influence on the methods and costs of mining.

The deposit lies at a slight angle to Gastineau channel. The hanging-wall of the Treadwell deposit is distant 1,000 ft. from the present shore-line, while that of the Ready Bullion lies for the most part under tidal water.

According to Prof. Becker, who undoubtedly had an excellent opportunity of studying the formation and character of the ore-bodies, this deposit represents the upper portion or "feather-edge" of an intrusion of sodium syenite, or, more logically, an albite diorite, which owing to peculiar conditions has been decomposed and silicified by solfataric or hydrothermal action, causing a concentration of gold in the native state and in the pyrites, sufficient to make it profitable mining. The country-rock in this locality is a carbonaceous slate, in which the original lines of bedding and principal cleavage coincide. It can be traced for miles both to the north and south. Its strike is N. 50° W., being practically the same as the ore-bodies and of Gastineau channel, and coinciding with one of the two fissure-systems which affect nearly the whole of the Alexander archipelago.

Following the deposition of the slate came the intrusion of the syenite. This dike is extremely irregular in shape. In the Treadwell mine it has a width of 420 ft.; going southeast it narrows down to a mere stringer until the Mexican mine is reached, where it again swells out to a width of 150 ft. From

the Mexican to the Ready Bullion, a distance of 2,500 ft., it is traced with difficulty on account of faulting and the broken character of the country. At the Ready Bullion it again attains a width of 300 ft.

The next intrusion, that of the gabbro, was of fundamental importance in the formation of the ore. Accompanying and following this intrusion came the mineralization of the syenite and the deposition of the ore. The dike of gabbro forms the hanging-wall of the deposit, and varies in width from 8 ft. in the Mexican to several hundred at the Ready Bullion. In places the gabbro is badly shattered and contains numerous stringers of quartz; but these are mostly barren, and nowhere is the gabbro converted into ore. This dike can be traced the full length of Douglas island.

The third and last of the intrusions was what Becker has called an analcite-basalt. This strikes N. 20° W., cutting the slate, syenite and gabbro, and dips to the west at an angle of 86°. Preceding and during this eruption, the country-rock was evidently badly shattered, and the fissures thus formed became filled with auriferous material. In no place does the basalt dike contain ore. In the upper levels it is quite continuous, but on the lower levels each dike has split into several smaller ones, which in places enclose fragments of the syenite.

In the Treadwell, Seven-Hundred-Foot and Ready Bullion mines, and in the west end of the Mexican, the ore consists of the mineralized syenite. This ore is readily divisible into two distinct varieties. The one consists of stringers of quartz and calcite, occupying the spaces formed by the crushing of the syenite; the syenite itself being somewhat mineralized along the planes of fracture. The other consists of syenite which has been crushed and broken by dynamic agencies, and the fragments thus formed have been saturated by the mineral-bearing solutions.

In the east end of the Mexican mine,—particularly in the lower levels,—the ore consists of auriferous quartz and calcite, occupying lenticular spaces in the black slate formed by its yielding to pressure along its cleavage-planes. The value of the ore can be readily determined by the quantity of pyrite contained and its distribution,—whether uniformly or in bunches,—the latter being the best. In the upper levels the best ore lies

next to the black slate foot-wall, but in the lower levels it has traveled toward the hanging-wall, and the space occupied by good ore above is here replaced by practically barren syenite.

### MINING.

*Shafts.*—The Treadwell, “700-Foot” and Mexican mines are opened by vertical shafts, but the Ready Bullion by an incline. In every instance the ground is of such a nature, except near the surface, that timbering would be unnecessary were it not required as a support for the skip-guides, ladders, pipes, etc.

TABLE I.—*Dimensions of Shafts.*

Name of Mine.	No. of Comp'ts.	Vert. or Incline.	Size of Timber.	Gal. Water Per Min.
Treadwell.....	4	Vert.	8'' x 8''	50
Mexican.....	3	Vert.	8'' x 8''	20
Ready Bullion	3	Incl.	8'' x 8''	80
700-Foot.....	3	Vert.	8'' x 8''	50

Table I. gives the size of the main working-shafts. All of these are rectangular, and are sunk by machine-drilling. Each mine has in addition from one to three smaller shafts, sunk for prospecting, which vary in depth from 150 to 400 ft. With the exception of the No. 1 Shaft of the Treadwell, they are not used for handling ore.

All shaft-sinking is done by contract, and if possible the contract for all work on the lowest level and for the shaft is let to the same parties. These contracts include the sinking, hoisting from the shaft, driving of drifts, trammings, etc., or, in other words, the company furnishes steam, air and tools, and there its liability ends. Two machine-drills are usually employed for sinking. Two sizes have been tried, and the larger, or the new Ingersoll machine—diameter of cylinder  $3\frac{1}{2}$  in.,—has been found superior to the 3.25-in. machine formerly used. The rock is quite hard, particularly on the hanging-wall side of the deposit, and it has been found that the number of feet drilled with the large machine is fully 30 per cent. more than with the smaller one.

Both 10- and 8-hour shifts were tried, and the best progress made with the former; consequently, all shafts, as well as other

underground work, are done by 10-hour shifts. The center-cut system of drilling is employed. The cut is first drilled, blasted and mucked out, then the other holes are drilled and blasted and the dirt removed in the usual manner; save that the end-holes are reserved, and fired with the cut-holes of the following round.

Although the shafts vary from 700 to 900 ft. below sea-level, and one of them is under the channel, very little water is encountered in sinking. Most of the water in the mines comes from the surface, and is caught up and pumped out, usually from the first level. The water underground does not average over 50 gallons per minute, and this is handled either by the skips or by No. 5 Cameron sinking-pumps.

Cross-head and buckets are used for sinking. The bucket is flat-bottomed, with the sides projecting 2 in., thus giving it a more secure base and permitting it to remain upright on an uneven surface while being filled.

The question of light at the bottom of a shaft has always been a vexatious one, owing to the water and blasting; but this question has been solved by using a group of three 36-c. p. electric lamps in the place of torches or candles. The wire is brought down the shaft to some convenient point, where a small reel is placed with which the lamps can be lowered or raised at will. It has been found that the best wire to use is the ordinary lamp-cord, wrapped with heavy canvas and given a good coat of P. & B. paint.

At the Ready Bullion mine the ore-body lies for the most part under water. For this reason a vertical shaft would be impracticable; therefore an incline shaft was started at a point 160 ft. from the foot-wall and sunk at an angle of  $47^{\circ} 53'$  from the horizontal. To protect the underground workings from the sea, a pillar 300 ft. thick was left in place above the highest workings in the main ore-body of the mine. The ore was cut at a distance of 728 ft. along the incline, and the shaft continued to the 750-ft. level. In the meantime the foot-wall had become much steeper, and it was evident that if the shaft was continued at the same angle, it would soon be out of the ore. The shaft was then gradually steepened until at the 900-ft. level it has an inclination of  $80^{\circ}$ , and the shaft continues at this angle to the next lower level.

*Stations and Ore-Bins.*—Formerly, it was the custom to open up a level every 110 ft.; but below the 440-ft. level the distance between levels in the Treadwell will be 150 ft. from now on. By this method a large development-expense will be avoided.

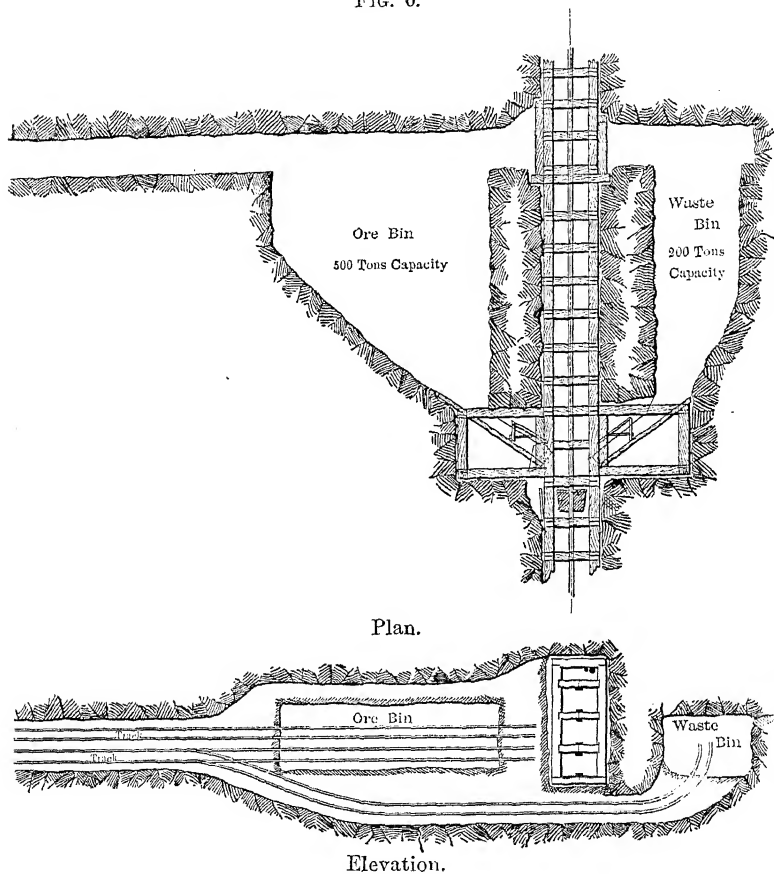
At each level a station is cut out, the width of the shaft, from 40 to 60 ft. long and with an average height of 8 ft. In the Treadwell the main cross-cuts run parallel to the wall-plates of the shaft, and as far as possible it is aimed to have the station on the side opposite from that toward which the skips dump. When the station-level is reached in timbering the shaft, a station-set is put in. This set is 14 ft. high, and is tied in two directions by iron bolts. To protect the front wall-plate that serves for the sill of the station, a stull 10 by 14 in. is put in between the wall-plate and the edge of the station, and the two securely bolted together. In cutting out the station, a drift is run from the shaft a distance of 25 ft. It varies in height from 15 ft. in front to 7 ft. at the back, and is the width of the shaft. The main cross-cut is then started at right-angles to the station-drift. For the first 80 to 100 ft. this cross-cut has a width of 20 ft., and from thence to the footwall it is 12 ft. wide. At the hanging-wall end of the cross-cut, a station is cut for the winding-engines that operate the tail-rope haulage; and directly opposite the sinking-compartment, on alternate levels, a station is cut for the sinking-hoist. Beneath the floor of each station an ore-bin is cut out with a capacity of from 500 to 1,500 tons, according to the quantity of ore to be handled. Experience has shown that it is very convenient to have large storage capacity at the stations, and when the ore-bin is in the ore it is just as economical to cut a large as a small one.

In the Mexican and "700-Foot" mines, the main cross-cuts start directly from the shaft, and as a consequence the station-cutting is much simplified. In other respects they follow the same system as the Treadwell, with the exception that the ore-bins are smaller, and in the Mexican two bins are cut,—one for ore in front of the shaft, and one for waste, cut on the opposite side of the shaft and reached by an extension of the main cross-cut. (See Fig. 6).

In the Ready Bullion mine the above methods are used, but

are changed in so far as to make them applicable to an incline-shaft, with the further exception that the main cross-cut runs to the hanging-wall. In cutting out the ore-bins two methods are employed. The old one consists of putting up a raise from the chute station to the level, and, after the chutes have been built, to cut out around the raise from the level of the cross-cut

FIG. 6.



SKETCH, SHOWING THE POSITION OF WASTE- AND ORE-BINS AT THE SHAFT OF THE MEXICAN MINE.

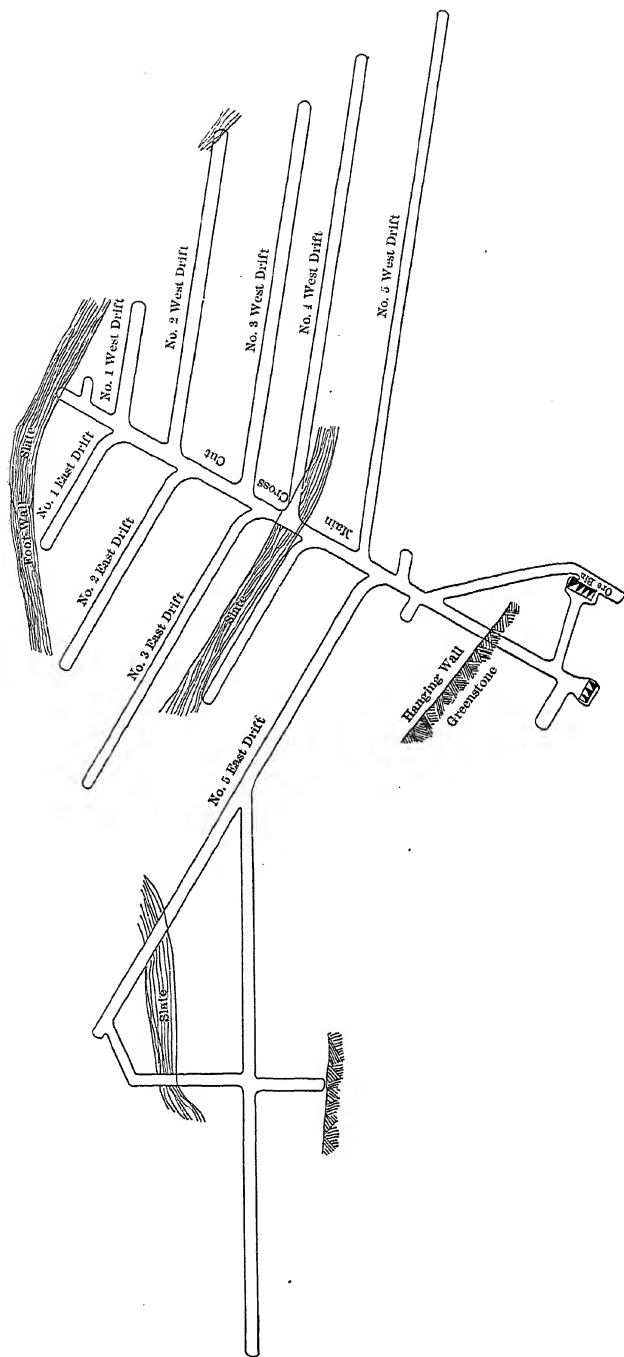
until the bin is of the desired dimensions. By this method the rock is broken into large pieces and has to be "bull-dozed" with powder and drawn out before the work of drilling can be continued. The second method is to put up a raise from the chute-station to the small drift from the shaft at the station level, then build in the chute and stope out a room the size of

the bin. The stope is carried to the level of the top of the main cross-cut, thus saving the expense of from 40 to 50 ft. of cross-cutting. At the same time it protects the timbers of the shaft from flying rock by leaving a small pillar between the ore-bin and the shaft, which can be removed when the bin is finished and the face of the cross-cut sufficiently far away to insure the shaft-timbers from being broken when the cut is blasted. The system of stoping is the same as that employed in the large stopes.

*Levels, Drifts, Cross-Cuts, Winzes, etc.*—When the ore-bin and station are completed, the main cross-cut is driven to the foot-wall. It thus serves the double purpose of determining how the level can be developed in the most economical way and of permitting a thorough sampling of the ore, so that a fair idea of its value can be obtained. The following rule is almost axiomatic in its applicability to the ore-bodies on the island: Start the first drift along the foot-wall, and keep it there. At various times the above rule has been ignored, and always with the same result. The ore-body is subject to abrupt changes in the strike and dip of the foot-wall. It is imperative that the ore be drawn from the stopes by gravity, and this cannot be applied to the ore along the foot-wall unless the chutes, and, consequently, the drifts, are kept as close to it as possible. Fig. 7 is a plan of the 220-ft. level of the Treadwell mine. On this level the slate-horse in the center of the deposit is quite small, and therefore more drifts than usual are required. On the lower levels, where the slate-horse is wider, it is left in place and forms a natural division between the hanging- and foot-wall portions of the ore-body. At intervals of 25 ft., raises are put up on alternate sides of both the main cross-cuts and drifts. These raises are 15 ft. high and are designed to accommodate the chutes for drawing the ore from the stopes. They are put up while the drift is run, and given a slope of  $60^{\circ}$  from the horizontal so that the ore will run freely in them. In the Treadwell, Ready Bullion and "700-Foot" mines, the drifts and chute-raises are in ore; but in the Mexican, on account of the flatness of the vein, they are run in the foot-wall slate, and the chute-raises put up to the ore at an average height of 20 ft. above the track. At the same time, as the main drift and the chute-raises are being run, a second drift, called the intermediate, is driven directly above the main drift and separated from



FIG. 7.



PLAN OF THE 220-FOOT LEVEL OF THE TREADWELL MINE.

TABLE II.—*Cost of Mining on Douglas Island, Alaska.*  
(Dollars Per Ton of Ore Milled.)

Name of Mine.	Machine-Drilling.										Rock-Breaking.															
	Tool-Sharpening.			Supplies.			Compressed Air.		Explosives.			Explosives.			Supplies.			Explosives.			Total Costs.					
	Labor.	Supplies.	Total.	Labor.	Merchandise.	Iron and Steel.	Repair-Labor.	Total.	Labor.	Steam-Power.	Supplies.	Total.	Powder.			Caps.	Fuse.	Supplies.	Labor.	Total.	Caps.	Powder.			Total.	
													70 Per Cent.	40 Per Cent.	Primers.							70 Per Cent.	40 Per Cent.	Labor.		
Treadwell .....	.0191	.0048	.0239	.2280	.0050	.0192	.....	.0242	.0332	.0233	.0233	.0030	.0018	.0073	.0556	.0042	.0825	.1543	.0370	.0042	.1233	.....	.0089	.1444	.3124	
Mexican .....	.0322	.0667	.0719	.2720	.0058	.0269	.....	.0267	.0081	.0572	.0245	.0848	.0012	.0176	.0822	.0066	.1068	.2137	.0337	.0067	.0029	.0916	.....	.0132	.1144	.3318
Ready Bullion .....	.0400	.0107	.0507	.2521	.0016	.0169	.....	.0174	.0029	.0882	.0064	.0975	.0012	.0154	.0709	.0068	.0965	.1439	.0041	.0064	.0029	.0874	.....	.0148	.1115	.2595

Name of Mine.	Tramming.				Hoisting.			Explosives.			Illuminants.			Total Costs.																				
	Labor.	Steam-Power.	Horses.	Supplies.	Total.	Labor.	Steam-Power.	Total.	Labor.	Candles.	Oil.	Electric Light.		Total.	Labor.	Supplies.	Iron and Steel.	Foundry.	Electric Light.	Hauling Sup.	Repairs.	Assay.	Power.	Personal.	Powder.	General.	Total.							
												Labor.	Supplies.																					
Treadwell .....	.0232	.....	.0042	.0055	.0335	.0170	.0023	.0233	.0233	.0486	.0141	.0100	.0060	.1312	.0656	.2269	.0010	.0672	.0002	.0001	.0085	.4706	.0504	.0360	.0012	.0003	.0021	.0080	.0015	.0536	.0089	.1966	.0017	0.8990
Mexican .....	.0185	.....	.0028	.0066	.0274	.0448	.0014	.0572	.1034	.0189	.0093	.0041	.1091	.0822	.2237	.0006	.0030	.0010	.0003	.0149	.6485	.0419	.0348	.0008	.0013	.0022	.0102	.0027	.1144	.0022	.0123	.0123	1.0390	
Ready Bullion .....	.0325	.....	.....	.0035	.0360	.0194	.0048	.0682	.1124	.0211	.0091	.0041	.1028	.0769	.2080	.0004	.0092	.0008	.0104	.5671	.0815	.0357	.0006	.0008	.0016	.0057	.0024	.1768	.....	.1787	.....	.....	0.9954	

NOTE.—The costs for the Treadwell mine are from May 31, 1901, to May 31, 1902, and for the Mexican and Ready Bullion mines, from December 31, 1901, to December 31, 1902.

o Tons milled: Treadwell, 682,893 tons; Mexican, 207,455 tons; Ready Bullion, 226,522 tons.

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its back by a pillar of rock 10 ft. thick. This drift is the same in size as the lower one, and is so driven that it connects with the top of each chute-raise as it progresses.

At the ends of the main cross-cuts, and at intervals varying from 200 to 500 ft. along the deposit, the different levels are connected by winzes. These winzes are used as man-ways and as a medium of ventilation. It might be well to add that they are always raised from the lower level, and not sunk from above unless circumstances absolutely require it. While running the main drifts and cross-cuts close attention is paid to the grade. The standard grade in all the mines is 0.5 per cent. This grade favors the loaded car going to the station, while it does not retard it too much on its return trip. Cross-cuts are used for connecting the various drifts and for prospecting. They are the same in size as the drifts. When used for the first purpose they are at the level of the drift, but when the second object is the incentive, they are usually driven from the level of the intermediate, so that the broken rock can be stored and handled through chutes. The usual size for drifts, cross-cuts and intermediates is 10 by 7 ft. in the clear, and for raises 6 by 8 ft. in the clear, no timber being used.

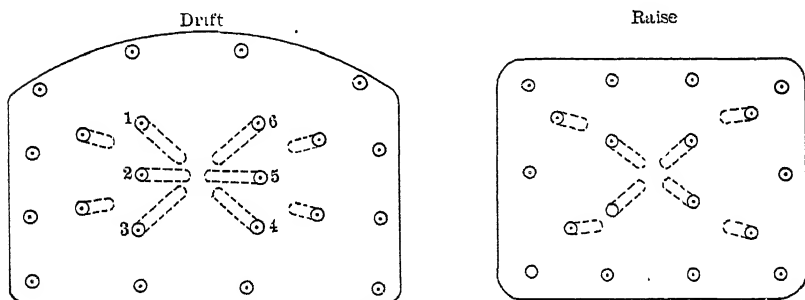
As a usual thing the rock is quite uniform in texture, and without seams or slips to break to. This makes it extremely difficult for the holes to break to the bottom. The following sketches (Fig. 8) show the scheme of drilling a round in a drift and a raise, which has resulted from several years' trials.

All work is done by machine-drills, two sizes being used. For drifting and cross-cutting the New Ingersoll, cylinder diameter  $3\frac{5}{8}$  in., is very satisfactory; and for raising, the Ingersoll-Sergeant, cylinder diameter  $3\frac{1}{4}$  in., is preferable, on account of its lightness and durability.

The cut-holes from No. 1 to No. 6 are drilled 7 ft. deep, and are so pointed that they come to a common apex at the center. The next row of holes, called the relievers, are drilled 6 ft. deep and given a slight rake toward the center. The outside holes, or trimmers, are also drilled 5 ft. deep, but are given a small outward angle. The cut is first blasted with No. 1, or 70 per cent, powder; the relievers and trimmers are then blasted separately, No. 2, or 40 per cent, powder being used in each case. The position of the blasting-holes, both in the drift and in the

*Stopping Surface-Pits.*—By reference to the table on page 384, it will be seen that considerably more than 75 per cent of the ore mined from the Treadwell has come from the open- or surface-pits. From now on, the percentage from the underground workings will become greater each year. The main open-pit, or "Glory Hole," has reached a depth of 220 ft. below the adit level and 450 ft. from the surface, with a maximum width of 420 ft. and a length of 1,700 ft. Owing to the large slides of waste rock from the foot-wall, and the necessity of having a secure pillar of rock at the 220-ft. level to protect the underground workings from surface-water, it is impracticable to carry the pits to a greater depth. In the other mines, at present, very little ore is being taken from the surface, but at the beginning, all of them depended on the open-pits for ore on

FIG. 8.



POSITION OF BLASTING HOLES.

which to start their mills. When a pit is to be opened, a raise is put up from the nearest level and connected with the surface. This raise is started from the intermediate drift, in general directly over a chute-raise. The chutes on each side then serve as a man-way for the raise in course of erection, and the broken rock is drawn off through the middle chute-raise into cars. When the raise has been connected, machine-drills are put to work cutting out a small stoep at the bottom. This raise when finished has the shape of an hour-glass, the top being formed by the open-pit and the bottom by a stoep, covering three chutes and from 20 to 30 ft. high, the two being joined by the raise. The object of cutting out the pit-raises in this manner is, first, to obtain chute-capacity in case of their being hung up by large pieces of rock or by blasting; and, second,

to afford an opportunity to break up any large piece of rock that may have been overlooked in the pit, which would stop up the chute unless it were broken to pieces small enough to pass through it.

Machine-drilling is seen at its best in these pits. The  $3\frac{1}{4}$ -in. diameter Ingersoll-Sergeant drills, set on tripods, are used in all the pits at present. The average number of feet drilled per machine in 10 hours is 36.35. The holes are drilled to an average depth of 12 ft., and each machine will break 69.69 tons of ore per shift of 10 hours. When the pits were smaller, and the difficulty of setting up was not so great as at present, the average number of feet drilled was much higher, and the breaking capacity of a machine-drill was from 150 to 200 tons of ore per shift of 10 hours. The pits are worked by drilling and blasting the ore from a series of benches or terraces around the chute-raise as a center, and when the ore is blasted the broken rock rolls down to the bottom. The small pieces are then broken by sledges, and the larger ones by placing sticks of powder on the surface of the rock, tamping with a little fine dirt, and blasting. For blasting holes, No. 2, or 40 per cent, dynamite is used, while for "bull-dozing" No. 1, or 70 per cent, is the best.

When the rock has been broken to the required size, it is drawn off, through the raises and chutes described above, into cars. These cars are hauled to the station ore-bins by horses, or by endless-rope haulage, where they are dumped. The ore is then loaded into skips, hoisted to the surface, and handled in the usual manner.

*Underground Stopping.*—The future economic workings of the mines depends on no one factor more than on the success attending the carrying of the present system of stopping to the lower levels of the mines. The surface-pits are practically exhausted, and the value of the ore does not allow of timbering or any extensive method of filling. So far, the present method has proved applicable to the lowest levels, and I do not hesitate to say that it will be equally successful at any depth to which it may be desired to carry it.

It was explained above that the object of the intermediate drift is to open communication with the ore-chutes and to furnish a large facial area for the machine-drills to work upon, in

cutting out or under-cutting the ground-floor for the stopes. When the intermediate has advanced about 50 ft., the work of cutting out the stope is started. This consists of mining out a chamber 7 ft. high, from 150 to 300 ft. long, and with a width varying with the width of the ore-body. In the past it has been customary to cut out the stopes with a level floor, but experience has shown that it is more economical to cut the floor so that it slopes from the parallel lines of chutes at an angle of about  $30^{\circ}$ . This does away with a large amount of shoveling, and the ore thus left is ultimately obtained through the stopes from the next lower level.

When the ground-floor has been cut out, the work of stoping upon the ore is immediately begun. The roof of the stope is arched, thus serving the double purpose of supporting the back and offering a better surface for the attack of the machine-drills. The ore is shot down in large, thin slabs, so that the shock of falling, combined with that of the blasting, breaks it up as much as possible. The pieces of rock too large to pass through the ore-chutes are broken by hand and "bull-dozed" with powder to the required size. When starting from the floor, the machine-drills cut out a trench along the center of the back to form the arch, its height varying with the character of the rock. Two sizes of machine-drills are used: the  $3\frac{1}{4}$ -in. and  $3\frac{3}{4}$ -in. Ingersoll-Sergeant, and the holes are drilled to an average depth of 8 ft. A machine-stoping will drill an average of 28.69 ft. per shift of ten hours and break 34.96 tons of ore with the consumption of 12.53 lb. of No. 2 dynamite. The cost of breaking up the rock after it has been blasted down is a large item in the expense of stoping. One rock-breaker is usually required to each machine, and it takes 0.85 lb. of powder in "bull-dozing" for each ton of rock broken.

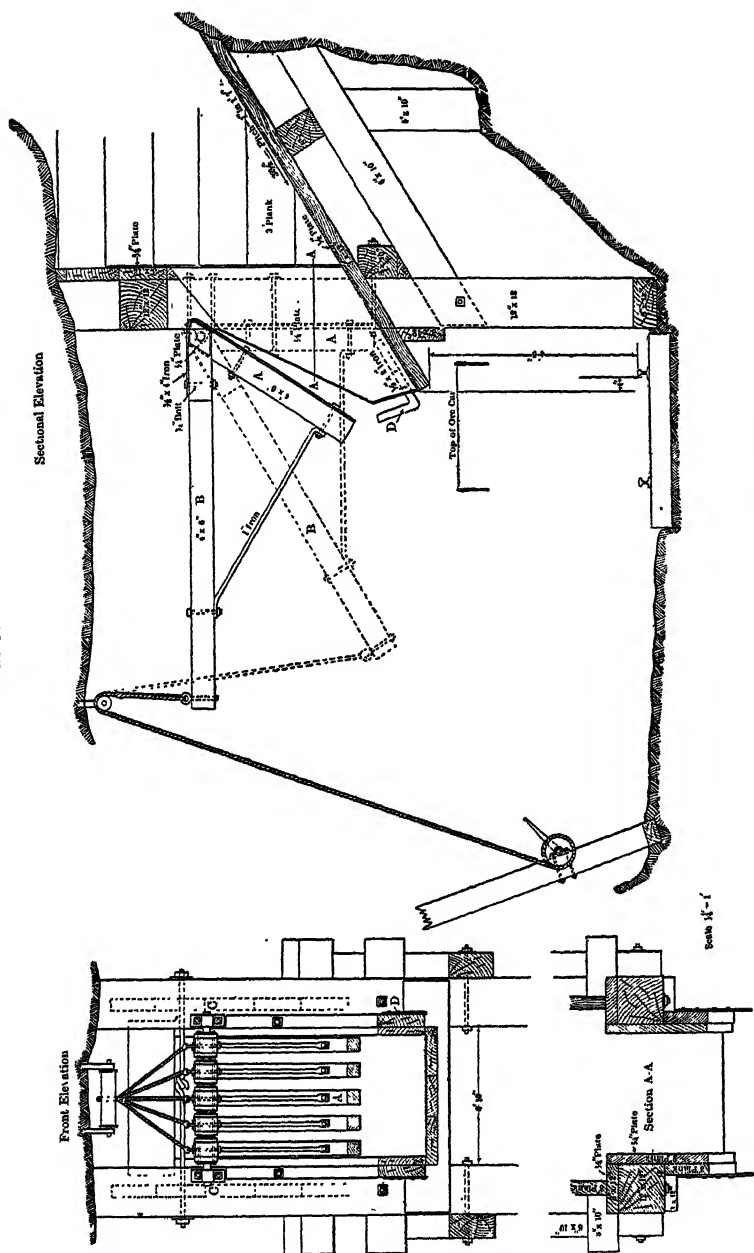
As no timber is used, it is compulsory that a sufficient quantity of broken ore be left in the stopes to form a solid working-floor for the miners. It has been found that one-third of the broken ore can be drawn off while the stope is being worked, and the surface of the broken ore kept within working distance of the back. In other words, by the above methods, two-thirds of the ore broken must be left in the stope, and cannot be drawn off until the stope is worked up to the next

higher level and finished. In the Treadwell and Ready Bullion mines the slate-horse forms a natural division between the stopes of the north and the south ore-bodies. The walls of the ore-body are supported by vertical pillars, or ribs, 15 ft. thick, and from 200 to 300 ft. apart. For means of communication and ventilation, man-way raises are put up in these pillars and connected with the levels. At intervals of 25 ft., short drifts are run in opposite directions from the man-way raise; so that, as the working-floor of the stope advances, each of them is used successively when the workings connect with the main raise, and in turn abandoned and closed up as connection is made with the next higher one. The levels are protected by horizontal pillars from 20 to 30 ft. thick. Heretofore these pillars have been left at each level, but from now on only the pillars at every other level will be left in place; yet even with this saving, fully 20 per cent of the ore must remain in the mine in the shape of pillars and ribs to support the ground and prevent caving.

*Chutes, Drilling, etc.*—Fig. 9 shows the principal features of the so-called finger-chute which is used throughout the island when a large amount of comparatively coarse rock is to be handled. The fingers are held in place by the weight of an arm, B, and are separately hung from the rod, C C, so that the motion of each is independent of that of the others. For the purpose of raising the fingers so that the ore can be drawn, to the top of each finger is fastened a piece of rope. These pieces are brought to a common center and fastened to the main hoisting-rope, which passes over a pulley in the top of the drift and thence down to a small windlass. When ore is to be drawn, the fingers are raised to the height necessary to allow the ore to pass. When enough has been drawn, the fingers are released and fall into position, stopping the flow. If a large piece is caught on the lip of the chute, it will hold only the finger or fingers that come in contact with it, while the others drop to their normal position. To prevent fine dirt dripping into the roadway, a tail-board is put across the mouth of the chute and held in position by the angle-irons, D.

The cost of putting up chute-raise, cutting-station, and installing finger-chute complete, is as follows:

Fig. 9.



STANDARD FINGER-CHUTE, TREADWELL MINE.



12 ft. of raise at \$9 per ft.,	\$108.00
80 cars of dirt at 20c.,	16.00
Cutting chute-station for windlass,	40.00
40 cars of dirt at 20c.,	8.00
Building finger-chute,	17.58
Lumber, 700 ft. B.M., at \$12.50,	8.75
Iron and steel,	40.00
Blacksmith-work,	7.00
Rope,	2.00
Total cost,	<u>\$247.33</u>

The principal advantages gained by using the finger-chutes are dispatch in loading and freedom from blocked chutes and from spilling ore on the tracks. Their cost is prohibitive except where a large amount of coarse, dry rock is to be handled, and in the Ready Bullion and Mexican mines they have been in a great measure superseded by the common board-chute, which costs very much less. In the Treadwell they alone are used, and, as stated above, they are being put in at intervals of 20 and 15 ft., instead of 30 ft., as heretofore. The additional expense of putting in finger-chutes every 15 ft., instead of every 30 ft., is \$8.33 per foot of drift. The saving in shoveling in a stope 60 ft. wide would be \$12 per foot, making a saving of \$3.67 by additional chutes; to which must be added the advantage of not losing time by chutes getting blocked. With chutes 15 ft. apart, the train can move with brief stops and be practically drawing from the same part of the stope. The blocked chute can be freed by one man and not delay the train more than 10 minutes. Also, in case the back of the stope should become so bad that men could not get in to work the ore into chutes, more than twice as much could be drawn from the stope with chutes 15 ft. apart.

A modification of the finger-chute is used for the skip- and bin-chutes. They are made with six fingers and are 6 in. wider than the ordinary chute. A trough is added to carry the ore from the lip to the edge of the shaft. It is 9 ft. long, 3 in. wide, with an inclination of  $36^{\circ}$ , so that the ore will run freely from the lip to the skip. Attached to the end of the trough is a hinged door made of 0.5-in. steel plate. When raised, it immediately and entirely stops the flow, and allows the trough to be filled in readiness for loading the next skip. It takes on an average 20 seconds to load a 4-ton skip and make everything clear for hoisting.

TABLE III.—*Duty of Machine-Drills on Douglas Island, Alaska.*  
“B.”

Mine.	Character of Work.	Average Number Feet Drilled Per Machine-Drill Per 10-Hour Shift.	Pounds Powder Used Per Machine-Drill Per 10-Hour Shift.	Average Number Tons Broken Per Machine-Drill Per 10-Hour Shift.	Average Costs Per Machine-Drill in 10 Hours.		
					Labor, Drilling.	Explosives.	Drill-Sharpening, Repairs, Supplies, Power, etc.
Treadwell mine.	Pits.....	36.35	15.89	69.69	.....	\$2.50	.....
	Stopes .....	28.69	12.53	34.96	.....	1.97	.....
	Cutting out.....	26.16	11.45	12.30	.....	1.80	.....
	Drifting.....	38.22	16.73	9.60	.....	2.63	.....
	Raising.....	34.18	14.95	7.60	.....	2.35	.....
	Shaft sinking....	31.12	13.65	9.28	.....	2.15	.....
Total....	.....	32.30	14.13	39.67	\$7.87	\$2.22	\$2.10
Mexican mine.	Pits.....	34.61	15.64	49.70	.....	\$2.46	.....
	Stopes.....	31.78	13.90	36.19	.....	2.26	.....
	Cutting out.....	31.40	13.74	18.40	.....	2.23	.....
	Drifting.....	40.83	17.86	11.40	.....	2.91	.....
	Raising.....	35.29	15.44	14.23	.....	2.51	.....
	Shaft sinking....	41.00	17.92	20.92	.....	2.92	.....
Total.....	.....	34.59	15.13	25.52	\$7.95	\$2.46	\$2.23
Ready Bullion mine.	Pits.....	39.63	17.34	197.06	.....	\$2.82	.....
	Stopes.....	29.85	12.96	38.04	.....	2.11	.....
	Cutting out.....	26.23	11.48	17.06	.....	1.87	.....
	Drifting.....	30.16	13.19	10.33	.....	2.15	.....
	Raising.....	26.44	11.57	6.96	.....	1.88	.....
	Shaft sinking....	26.80	11.72	15.21	.....	1.91	.....
Total.....	.....	28.58	12.49	28.19	\$7.03	\$2.03	\$3.02
700-Foot mine.	Pits.....	36.41	15.92	55.15	.....	\$2.59	.....
	Stopes.....	29.26	12.81	35.68	.....	2.09	.....
	Cutting out.....	26.00	11.37	12.40	.....	1.85	.....
	Drifting.....	38.00	17.12	15.64	.....	2.71	.....
	Raising.....	33.60	14.70	8.60	.....	2.39	.....
	Shaft sinking....	32.16	14.07	20.50	.....	2.29	.....
Total.....	.....	31.33	13.72	27.41	\$6.48	\$2.23	\$3.23
Average Totals..	.....	31.92	13.96	33.54	\$7.58	\$2.27	\$2.42

Table III. gives the duty of machine-drills in the different characters of work for which they are employed.

Under the heading “Pounds powder used per machine-drill per 10-hour shift” is given the actual amount of powder used for blasting the holes drilled, but it does not include powder used for bull-dozing, blasting chutes, etc. The cost of this powder is given under the head of explosives; it varies with

the different mines, and from month to month in the same mine, according to the varying expense of handling and sales to outside parties.

The "Labor, Drilling" cost given is the actual expense of machine-men and helpers. The men do a great deal of the barring-down and trimming, and the above cost will vary accordingly.

The last column under "Average cost per machine-drill in 10 hours" gives the cost of drill-sharpening, repairs, power, and all other costs not included under the heads of Labor, Drilling and Explosives.

*Tramming.*—At present there are three methods of tramming in use: by hand, with the help of horses, and by endless-rope haulage.

When the tramming is done by hand, one man loads a car, pushes it to the ore-bin, dumps it, and returns to the chute. Ordinarily there are several men tramming from the same chute, and a great deal of time is lost by awaiting their turn to load; and again at the bin there is another wait until the last car is dumped before starting on the return-trip.

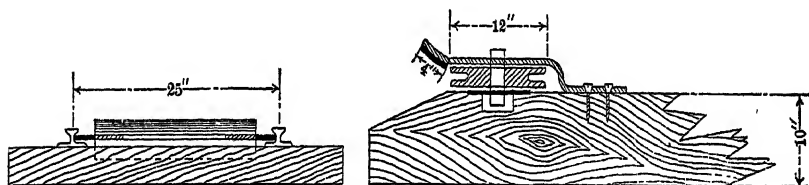
By using horses, the lost time is minimized by making up two or more trains of 8 cars each, so that when one train is dumping, another is loading and the third is *en route*.

While there is a marked saving in men by using horses, their first cost is considerable, a horse costing \$180 delivered; and their depreciation is another factor to be considered, for they are continually meeting with accidents, which either disable them permanently or lay them off at a cost of \$1 a day for maintenance.

By the use of mechanical haulage, both the lost time and cost per ton has been greatly reduced. The system is essentially the so-called "tail-rope" system in use in numerous places, but modified to make it conform with underground conditions. On the hanging-wall side of the ore-bin is situated a double-drum winding-engine, size of cylinders 7 in. by 10 in., with drums 2 ft. 8 in. in diameter. Set directly in front of and close to the engine are four posts. Two are designed to carry a sheave suspended on a horizontal axle, for guiding the upper rope and causing it to wind smoothly on the drum, while the remaining two support a roller which answers the same pur-

pose for the lower rope. From the drum to the point where the drifts branch out, the upper rope is supported by snatch-blocks suspended from the back of the drift and by sheaves at the ends of arms securely fastened to 10- by 10-in. posts. A horizontal sheave is placed at the point where the direction of the rope is changed to allow it to enter the drift. Since this sheave is subject to severe strains, it should be held rigidly in place by horizontal 10- by 10-in. pieces securely bolted to the 10- by 10-in. posts. From this point to the end of the drift the upper rope is carried by sheaves fastened to the posts of the finger-chutes, immediately under the protruding lip, where it will be out of the way and at the same time protected from blasting. The sheaves are inclined so that the greatest strain is at right-angles to the axle, and the rope is prevented from jumping out by pegs placed across the top of the sheave. The lower rope is kept in line by a series of horizontal sheaves fastened

Fig. 10.



LOWER ROPE SHEAVE AND GUARD.

to blocks, their number depending on the crookedness of the tunnel; and it is prevented from dragging on the ground by iron rollers placed between the tracks.

The lower sheaves (Fig. 10) are placed as near the track as possible, and are mounted on pieces of 10- by 10-in. timber securely braced against the side of the tunnel. To guide the rope into the sheave the front end of the block is beveled off to the height of the rail. On the top of the sheave is a piece of wrought-iron, bent as shown in cut, its object being to prevent the rope from jumping out and also to hold the axle in a vertical position. At the ends of the various drifts, or at convenient points in them, is placed a series of sheaves, or a single sheave as the case may be, to carry the end-loop of the rope. At first was tried a sheave mounted on a truck and fastened by clamps to the rails; but this proved a failure on account of the great strain, which pulled up the track and did other damage.

For purposes of signaling, two bare iron-wires are run the entire length of the tunnel. These wires are parallel and 4 in. apart. At the winding-engine they connect with a bell and signal-light, while the current is obtained from the electric-light circuit. The signals are given by placing an iron candlestick across the two wires or by means of a special portable signal device. As the wires are bare, signals can be given from any point, which is a great convenience in case of the train jumping the track or other accident. Two trains are used on each level, consisting of seven cars, each car holding 1.5 tons. While one train is discharging at the shaft ore-bin, the other is loading. The trains are run at a maximum speed of 800 ft. per minute, and at present their capacity is 750 tons of ore per shift, or 1,500 tons per day.

*Sampling, Maps and Assaying.*—Close attention is paid to sampling and recording the assay-value of the ore. As a drift, raise, cross-cut, or other development-work is in progress, a sample is taken after each round has been blasted. These samples are taken either by the shift-boss or the foreman, and their description and location are recorded on a special tag, enclosed with the sample in the sack.

At intervals of 15 ft., and closer if there is any doubt as to the value of the ore, lines of samples—each sample being 10 ft. long and varying with the nature of the ore—are taken across the back of the stopes at right-angles to the strike. These samples are taken by cutting trenches, usually 10 ft. long, 4 in. wide across the strike of the ore, and 5 ft. apart, for the entire length of the new work. A hand-sample is taken from each car at the ore-bins, and again at the crushers a grab-sample is taken by means of large dippers, before the ore goes to the mill.

The only samples taken in the mill are from the tailings and sulphides. The tailings-sample is taken either at the tail of the vanners by means of dippers, or at the end of the tailings-launders by means of automatic samplers. The sulphides are sampled by means of the ordinary grooved sampling-rod. When the mine-sample reaches the assay office it weighs from 50 to 150 lb.

The assay office is fitted with the usual grinding machinery, two double-muffle furnaces, pulp- and bullion-scales, etc. There

is an average of about one hundred samples assayed each day, and the returns from a sample are obtained the day after it is taken. All samples are assayed for gold only. In connection with the assay office are the retorting- and melting-rooms for refining the amalgam from the mills.

A complete set of maps is kept, showing in detail the underground and surface-workings of the mines, also the value and position of each sample taken and the quantity of broken ore and reserves.

*Labor.*—The nationality of the mine-laborers is shown in Table IV.

TABLE IV.—*Number and Percentage of Different Nationalities Working at the Mines on Douglas Island, Alaska.*

Nationalities.	Per Cent.	Treadwell.	Mexican.	Ready Bullion.	Total.
Americans.....	26.5	147	46	27	220
Norwegians, Swedes and Danes	25.	104	60	41	205
Scotch.....	1.	4	2	5	11
Austrians and Slavonians .....	24.5	113	29	58	200
Irish.....	3.5	22	5	2	29
Germans.....	3.	13	7	4	24
French.....	.....	2	.....	.....	2
English.....	2.5	16	2	2	20
Italians.....	3.	6	8	9	23
Finns.....	5.	17	20	3	40
Russians.....	1.	5	3	.....	8
Turks.....	.....	1	.....	.....	1
Japanese.....	4.	17	15	.....	32
Indians.....	1.	6	2	1	9
Totals .....	100.	473	199	152	824

On account of the system adopted for working the mines, due to the character of the walls and vein-material, it is necessary to employ only skilled labor in the shafts, drifts, raises, etc.

About 60 per cent. of the machine-men and helpers on the island came as laborers and have learned their trade here. They are preferred by the foremen, and seem on the average to break more rock than miners who have gained their experience elsewhere.

Following are the rates of wages paid the different classes of labor employed :

Machine-drillers, . . . .	{ \$2.50 per day in summer, \$3.00 per day in winter ; with board and lodging.
Machine-helpers, . . . .	\$2.25 per day, with board and lodging.

Mine-laborers, . . .	\$2.00 per day, with board and lodging.
Mine-laborers (Indians), . .	\$2.00 per day, without board and lodging.
Blacksmiths, . . .	\$4.00 per day, with board and lodging.
Tool-Sharpener, . . .	\$3.50 per day, with board and lodging.
Blacksmith's helpers, . .	\$2.00 per day, with board and lodging.
Machinists and helpers in machine-shop, . . .	From \$2.00 to \$6.00 per day, with board and lodging.

Labor of all kinds is plentiful, particularly so in the winter months.

*Machinery.*—Tables V., VI. and VII. show in detail the size of the principal hoisting-engines, compressors and water-wheels. Besides the above there are several hoisting-engines used underground, for sinking, tramming, etc.

All the machinery, with the exception of the hoisting-engines, is arranged so that at any time when there is sufficient water the steam can be shut off, wholly or in part, and the power be furnished by water-wheels. These wheels are either direct or rope-connected.

The steam at all the mines, with the exception of the Mexican mine where five return-tubular boilers are used, is supplied by Heine safety boilers arranged in units to suit the needs of the various mines.

Situated as the mines are, far from their source of supply, it is but natural that they support a well-equipped machine-shop and foundry, where all repairs, mine-cars, drill-parts, etc., are made.

There are no wagon-roads. The steamers deliver all supplies and machinery on the company's wharf, where it is reloaded on cars and delivered to all the principal points about the mines. For the surface-work, two 7-ton mine-locomotives, made by the Baldwin Locomotive Works, with necessary coal- and flat-cars, are used.

#### MILLING.

The character of the ores on Douglas island is peculiarly adapted to the simple methods of extraction in use. As explained above, the gold is contained in an altered syenite in the form of free gold, and in the sulphides, the principal gold-bearing minerals being iron pyrites, arsenopyrite, molybdenite and calcite. The ore on the surface has been subject to little oxidizing action, and, perhaps, that on the lowest level is even

TABLE V.—Dimensions of Principal Air-Compressors on Douglas Island.

Name of Mine.	Make and Character of Compressor.	Steam-Engd.										Pelton Water-Wheels.					Air-Engd.														
		Diameter of Cylinders in Inches.	Piston-Stroke in Inches.	Piston-Speed Per Minute.	Cylinder Area in Sq. Inches.	Mean Effective Pressure in Pounds.	Boiler-Pressure in Pounds.	Indicated Horse-Power.	Low.	High.	Low.	High.	Diameter of Wheel.	Diameter of Nozzle.	Head of Water.	Gauge-Pressure.	Number of Wheels.	Remarks.	Number of Cylinders.	Diameter of Air-Cylinders, Inches.	Area of Air-Cylinders, Sq. Inches.	Length of Stroke in Inches.	Inlet.	Outlet.	Diameter of Valves, Inches.	Area of Valves, Sq. Inches.	Possible Speed Under Pressure, Rev. Per Min.	Revolution Run.	Maximum Drills Obtained.	At Air-Pressure of Pounds.	
Ready Bullion.	Reidler duplex.	34	20	36	36	372	372	907.92	314.16	14	43.3	100	100	143.3	152	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Mexican.	Reidler duplex.	30	18	30	30	850	850	706.86	254.46	14.2	49	90	90	106	132	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Treadwell.	Ingersoll single cylinder.	.....	.....	.....	.....	.....	.....	907.92	314.16	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Treadwell.	Reidler duplex.	34	20	36	36	372	372	907.92	314.16	14.0	43.3	100	100	148 3/4	152.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....



TABLE VI.—*Dimensions of Hoisting-Engines on Douglas Island, Alaska.*

Name of Mine.	Treadwell.			Mexican.	700-Foot.	Ready Bullion.
Location of Hoists.	New Shaft.	Old Shaft.	Man-Way.	New Shaft.	Main Shaft.	Main Shaft.
Number of cylinders..	2	2	2	2	2	2
Diameter of cylinders in inches.....	15	12	10	14	12	12
Piston-stroke in inches	24	16	12	18	16	16
Piston-speed per min...	600	400	300	450	400	400
Cylinder area in sq. in.	176.71	113.09	78.54	153.93	113.09	113.09
Mean effective pressure in pounds.....	60	60	60	75	60	60
Boiler pressure,pounds	100	100	100	100	100	100
Indicated horse-power of hoist.....	385.2	164.4	84.4	314	164	164

TABLE VII.—*Dimensions of Mill-Engines and Water-Wheels,  
Douglas Island, Alaska.*

[illegible]

TABLE VIII.—*Equipment of Mills and Cost of Milling on Douglas Island, Alaska.*  
Equipment of Mills.

Name of Mill.	Crushers.			Capacity of Ore Bins. Cubic Feet.	Stamps.			Feeders.		Mortars.		Amalgamating Plates.				Vanners.			Motive-Power.			Time.				Average in 24 Hrs. Crushed in 24 Hrs. Tons
	Number.	Size.	Make.		Number.	Weight.	Drops Per Minute.	Height of Drop.	Number.	Style.	Number.	Style.	Number of Chuck Blocks.	Number.	Grade Per Ft.	Area in Sq. Ft.	Number.	Style.	Size.	Kind.	Number.	Style.	Water-Wheels.	Per Cent. by Water.	Per Cent. by Steam.	
Treadwell "300."	2	"E."	Comet.	81,735	800	1020	98	8 1/2"	60	Suspended Challenge.	60	"67-A," F. and C.	60	1 1/2"	45	Frue.	4	Water.	93.19	.....	93.19	6.81	5.53			
Treadwell "240."	2	.....	Gates.	50,400	240	850	98	.....	48	Challenge.	48	Moran.	48	1 1/2"	45	Frue.	2	Steam and water.	51.64	40.19	91.83	8.17	4.86			
Mexican.....	1	.....	Gates.	81,660	120	1020	98	8 3/4"	24	Challenge.	24	"67," F. and C.	24	1 1/2"	45	Frue.	1	Steam and water.	41.64	36.42	78.06	21.94	5.73			
Ready Bullion.....	1	"E."	Comet.	49,380	120	1020	100	8 3/4"	24	Suspended Challenge.	24	"67-A," F. and C.	24	1 1/2"	45	Frue.	.....	Steam.	.....	.....	.....	97.82	97.82	2.18	5.34	
700-Foot Claim. ..	1	"E."	Comet.	33,150	100	1020	99	8 3/4"	20	Suspended Challenge.	20	"67-A," F. and C.	20	1 1/2"	45	Frue.	1	Steam and water.	92.22	4.88	97.10	2.90	5.69			

Cost of Milling (Dollars Per Ton of Ore) for Ten Months Ending September 15, 1902.

Name of Mill.	Crushing.		Concentration.			Sulphurets.				Total of All Expenses.														
	Labor.	Supplies.	Total.	Labor.		Total.	Labor	Supplies.	Wharf.	Car.	Assay.	Total.	Labor.	Supplies.	Iron and Steel.	Foundry.	Electric Ligh.	Hauling Sup- plies.	Repair.	Assay.	Power.	Legal.	Total.	
				Concen- trators.	Sulphurets.																			
Treadwell, "300" .....	.0082	.0046	.0128	.0091	.0042	.0095	.0099	.0147	.0068	.0007	.0005	.0003	.0015	.0007	.0144	.0211	.0109	.0001	.0003	.0057	.0004	.0061	.0015	.1206
Treadwell, "240" .....	.0095	.0040	.0135	.0133	.0047	.0029	.0099	.0308	.0047	.0007	.0005	.0003	.0015	.0007	.0720	.0376	.0058	.0010	.0006	.0106	.0016	.0658	.0032	.2040
Mexican .....	.0108	.0007	.0116	.0095	.0080	.0060	.0090	.0325	.0052	.0003	.0005	.0103	.0006	.0003	.0720	.0523	.0220	.0121	.0027	.0009	.0001	.0017	.0800	.2362
Ready Bullion .....	.0261	.0019	.0280	.0147	.0072	.0177	.0185	.0586	.0087	.0002	.0052	.0006	.0146	.0005	.0884	.0360	.0362	.0135	.0009	.0012	.0009	.0020	.0035	.3103
700-Foot Claim.....	.0096	.0008	.0104	.0108	.0095	.0014	.0086	.0288	.0047	.0005	.0005	.0106	.0005	.0005	.0712	.0285	.0165	.0021	.0002	.0004	.0014	.0575	.0669	.1981

more free-milling than that in the surface-pits. By reference to Table IX. it will be seen that 48.04 per cent of the gold is caught on the plates by amalgamation, and the balance, or 51.96 per cent, is contained in the sulphurets and tailings. Table IX. shows very plainly the effect of coarse crushing on the percentage of extraction in the various places.

*Crushers.*—The crushers are located in the head-frames of the various mines, and are of the gyratory type. When the ore is hoisted out of the mine it is spilled by self-dumping skips on to a grizzly formed by 1-in. by 10-ft. pieces of iron, bolted together by 1-in. iron bars, and placed 2 in. apart by disc-shaped pieces of cast-iron. The over-size from the grizzlies goes direct to the crushers, and the under-size passes through and falls into the ore-bins situated directly beneath the crushers.

Too much stress cannot be laid upon the great effect of efficient crushing as related to the duty and output of a stamp-mill. This is particularly true on the island, where the crushing capacity is in excess of the demand and where there is abundant water-power which costs practically nothing. During the past year the duty of the mills has been increased over 1 ton per stamp in 24 hours, and without a doubt, 50 per cent of this increase has been caused by setting the crushers to break the rock 20 per cent smaller than before. An efficient crushing-plant for mines similar to the Treadwell would consist of four Gates-crushers arranged in pairs, one above the other, the upper to be of such a size that they would receive rocks 18 by 36 in., and the lower to turn out a product not larger than 1.5 in. in diameter. The rock when hoisted would be dumped on grizzlies with 5-in. spacing between bars: the over-size going to the upper crushers and the under-size falling on a second grizzly with bars set 1.5 in. apart,—over-size going to the lower pair of crushers and under-size passing into the storage-bins. The product from the upper pair of coarse crushers to be spilled on a grizzly with bars 1.5 in. apart, the over-size going to the lower crusher and the under-size and crushed product from the lower crushers falling into the bin. If the above method were used it would do away with a great deal of the bull-doing and rock-breaking in the mines, making a very appreciable reduction in the cost of mining.

There are three different methods in use for conveying the ore from the crusher-bins to the mill ore-bins. At the Treadwell, small locomotives are used, drawing trains of six cars, each car holding 2.5 tons. At the Mexican, where the crusher and mill are practically under one roof, the 2.5-ton-cars are pushed by hand; while at the Ready Bullion the ore is handled by means of a gravity-tram operating a train of four 2.5-ton cars, the cars being returned by means of a small winding-engine located at the crusher-bin. The tracks from the crusher ore-bin are continued along the top of the mill ore-bins, so that the ore can be dumped directly from the cars into the bins.

In the 300-Treadwell and 240-stamp mills, the stamps are arranged back to back, and the bottoms of the bins are made in the shape of an inverted V, so that the ore will be equally divided and fed uniformly to the stamps on either side. In the other mills, where the stamps are arranged in a single row, the bottom of the ore-bins, from a point 8 ft. below the track, is given a slope of  $45^{\circ}$  to the open ore-chutes at the level of the cam-floor. The bins are double-boarded, and on the side next the stamps are lined with 0.25-in. steel-plate, to protect them from the scouring action of the rock. From the bins the ore is taken out by openings at the level of the cam-floor, and conveyed by chutes to the hoppers of the challenge-feeders. There is one chute and feeder for each five stamps. The 300-Treadwell, Ready Bullion, and 700-Foot mills are provided with the suspended challenge-feeders. These are preferable to the standard feeder, being more compact and very accessible for repairs. Both types are central feeders, the bumper-rod being placed next to the central stamp and guided in the usual way.

There are three different kinds of mortars in use on the island. The 300-Treadwell, Ready Bullion, and 700-Foot mills use the Fraser and Chalmers No. 67-A Type; the Mexican uses the Fraser and Chalmers No. 67, while the 240-Treadwell mill uses a special mortar made by Moran Bros., of Seattle, Wash. End- and side-liners are used in all the mills; and false bottoms are used, except in the 240-Treadwell mill, where the die rests on the bottom of the mortar. The false bottoms and liners are cast at the company's foundry, which does excellent work. The false bottoms in use consist of a piece of

cast-iron 3 in. thick, and of the size and shape of the flange-portion of the die. Their object is to protect the bottom of the mortar from excessive wear.

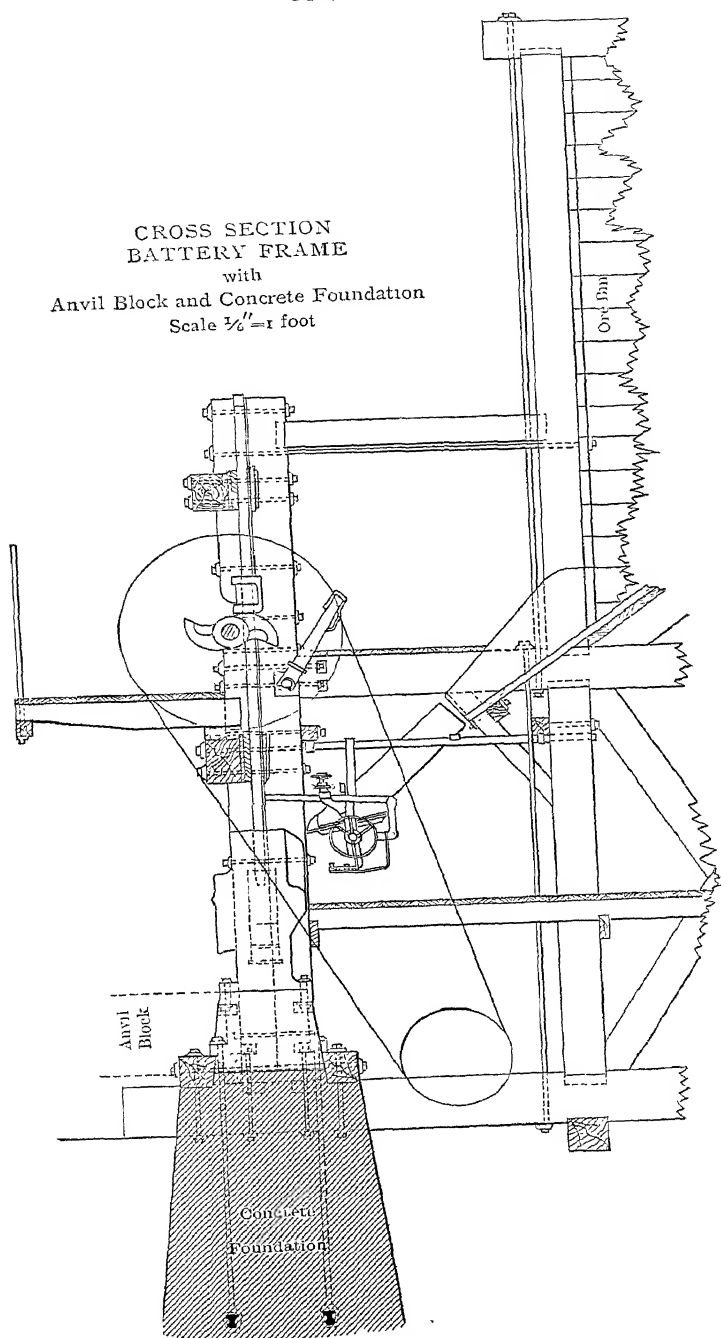
The stamps in the 300-Treadwell, Ready Bullion, Mexican, and 700-Foot mills weigh 1,020 lb., while those in the 240-Treadwell weigh when new 850 lb. The stems, tappets, boss-heads and shoes are joined in the usual manner. The Koppel shoe is used in all the mills. A shoe lasts 3 months and crushes 489 tons of ore, which means that 0.27 lb. of iron is consumed per ton of rock crushed. The dies are cast at the company's foundry and last on an average 4.49 months, crushing 732 tons and consuming 0.16 lb. of iron per ton crushed.

Interior views of the battery- and concentrator-floors of the 300-stamp mill at Douglas island are well shown in Figs. 12 and 13; and Fig. 14 shows the position of the Treadwell-Riedler compressor and the Pelton water-wheel, the latter having the cover removed.

*Battery-Foundations.*—There is no one part of a mill so essential to its efficiency as a good and lasting mortar-foundation. There are two kinds used on Douglas island. The Ready Bullion, 700-Foot, and the 300-Stamp mill at the Treadwell have concrete foundations, capped by cast-iron anvil-blocks, while the Mexican and the Treadwell 240-stamp mills have the ordinary wooden foundation.

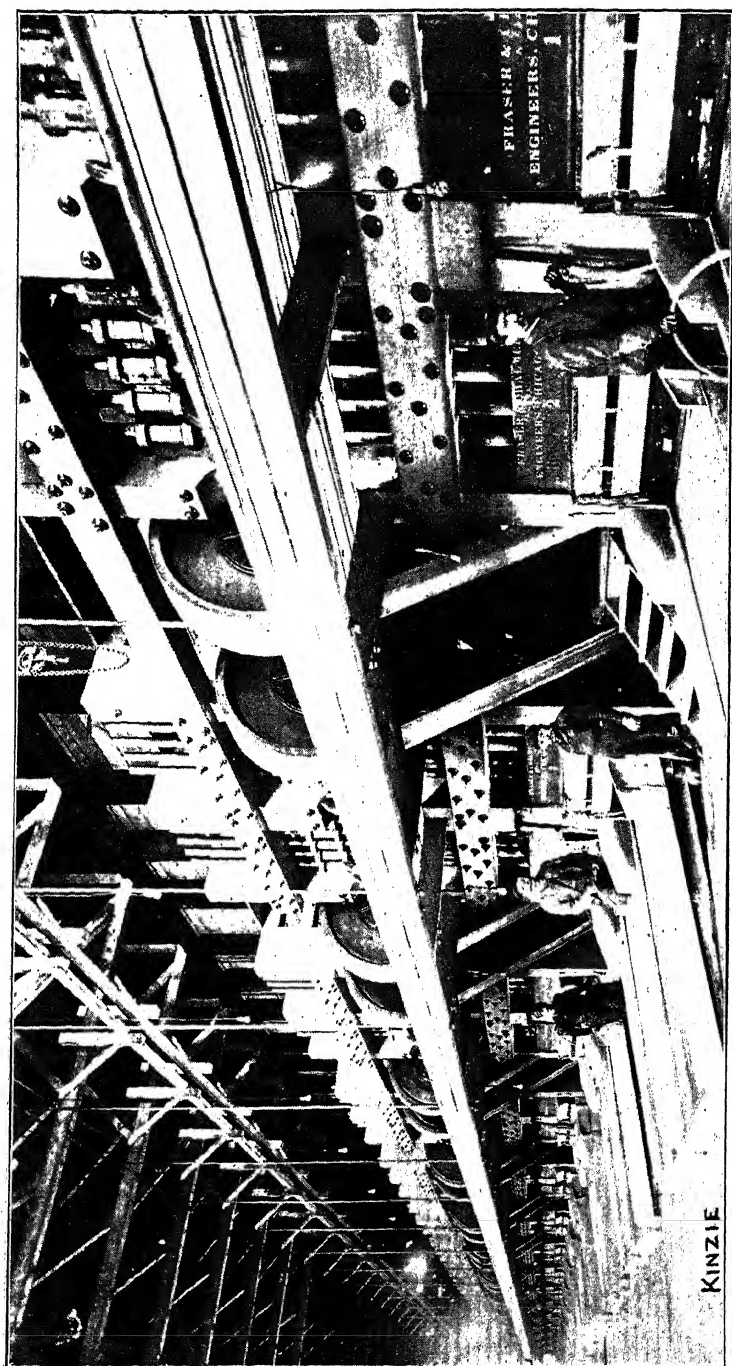
On account of the peculiar climatic conditions prevalent on Douglas island, the life of a wooden foundation made of the best Oregon or Douglas fir has been about six years. It then commences to rot, causing the mortar-bolts to loosen, which results in an uneven wearing of the top-surface of the block. The life of the mortar-block can be prolonged from six months to a year by planing down the top and leveling it up by placing strips of rubber-beltting between the old mortar-block and the mortar, but this is at best a make-shift. The rotting in every case has been confined to the surface, while the interior of the block remains sound. The blocks are built in the usual way, with four pieces of Oregon pine, 4 ft. 8 in. by 26 in., bolted together by iron bolts. The holes for the anchor-bolts are drilled into the mortar-block from above, at distances apart corresponding to those on the flange of the mortar. At a distance of 4 ft. below the top, holes are cut out to receive the

FIG. 11.



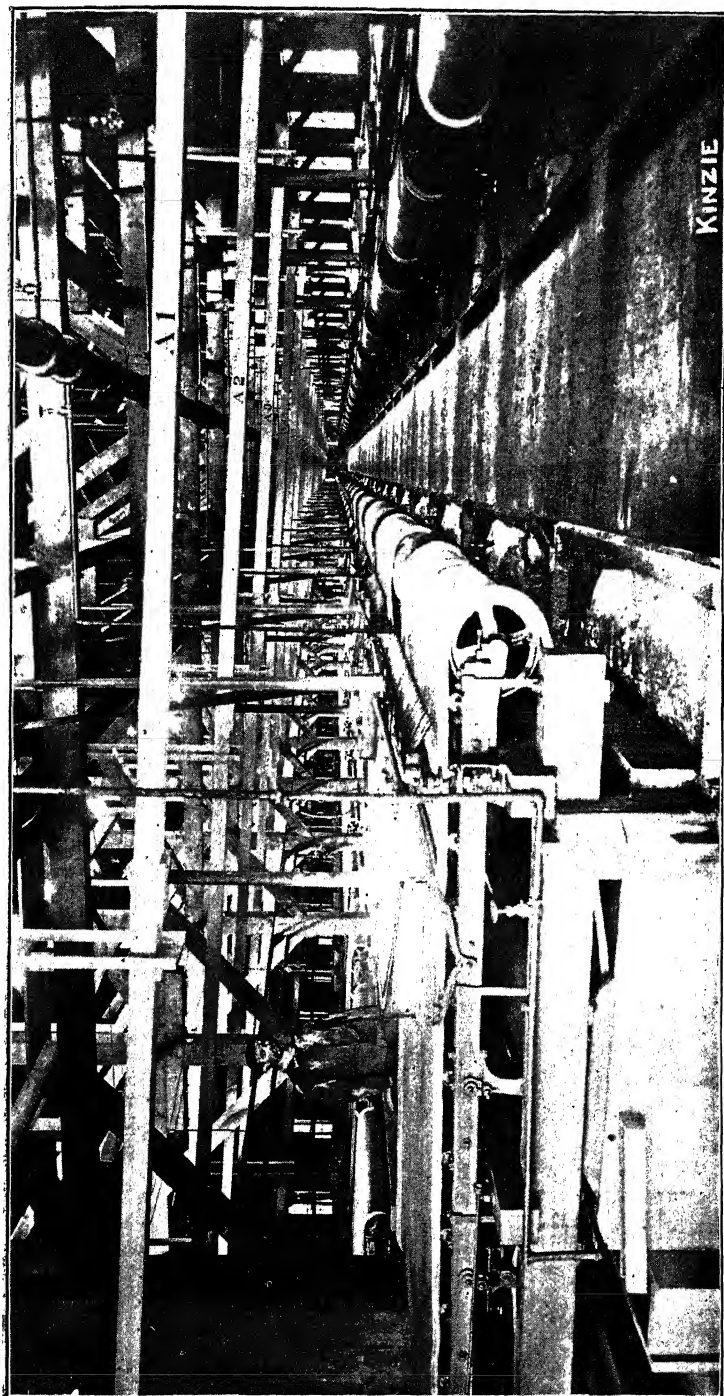
DETAILS OF BATTERY FRAME.

FIG. 12.



BATTERY FLOOR OF THE 300-STAMP-MILL AT DOUGLAS ISLAND.

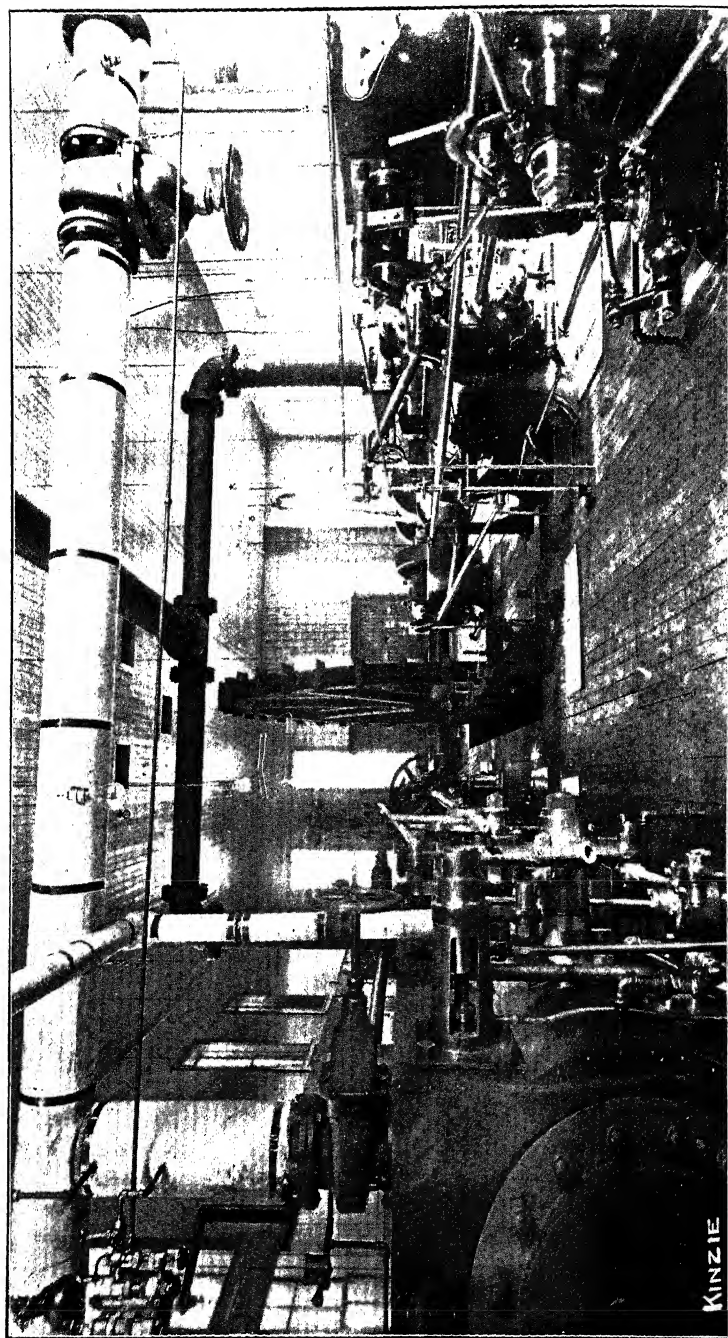
Fig. 18.



CONCENTRATOR FLOOR OF THE 300-STAMP-MILL AT DOUGLAS ISLAND.



Fig. 14.



TREADWELL-RIEDLER COMPRESSOR, SHOWING PELTON WATER-WHEEL WITH COVER REMOVED.

nuts and washers that secure the lower end of the bolts. It is evident that when the decay of the wood penetrates to, say, a depth of 4 in., there is no longer a secure fastening for the mortar, and the mortar-block becomes useless. If a 2-in. iron rod be passed through the ends of opposite mortar-bolts and through the block, the life of the foundation will be prolonged from two to four years. The best wooden block is one built up of 2-in. plank, bolted and nailed together, and the mortar-bolts fastened as above.

Fig. 11 shows the character of foundation used in the newer mills. Soon after the mills were started it was noticed that the edges of the concrete next to the mortar showed signs of crumbling. The mortars were then raised and a sheet of 0.25-in. rubber-belt inserted between the mortar and the anvil-block. This lessened the crumbling, but did not stop it, and during the third year, fully one-quarter of the mortar-blocks in the mill had their anchor-bolts broken off at the point of contact of the concrete and the anvil-block. In nearly every instance, on removing the anvil-block, it was found that when the foundation was installed, instead of surfacing up the concrete to the correct level, it was allowed to set; then cement was poured in between the anvil-block and the already hardened concrete to raise the foundation to its proper height and level. This caused a plane of weakness and consequent crumbling, resulting in the breaking. The above condition might be remedied in the first instance by properly finishing the concrete, and making it 8 in. wider on either side than the bottom of the anvil-block, as shown in Fig. 11. With the above exceptions, the concrete foundation with the anvil-block has answered the purpose for which it was designed, and with a few minor changes it is preferable to the wooden foundation under the existing conditions on Douglas island.

*Power.*—Starting with the 5-stamp mill of the Treadwell, erected in 1881, the number has been increased from time to time, until at present there are 880 stamps dropping on the island, and these are distributed in the various mills as shown in Table VIII. Table VII., on page 364, gives the dimensions of the mill-engines and water-wheels at Douglas island.

During the summer months there is sufficient water to run 760 of the 880 stamps, and during the winter there is always

enough to supply the batteries and vanners and run a portion of the mills.

This water-supply is obtained from a series of ditch-lines running along the mountain-sides and aggregating 18 miles in length. The main Treadwell ditch starts from a lake in the mountains 14 miles distant from the mines. It then follows the contour of the mountain-range, gathering the water from numerous small springs and streams, and delivers it to the penstocks at an elevation of 480 ft. above the mills, and at a distance of 1,500 ft. from them. On account of the heavy snow-fall and low temperature in winter, the ditches are covered throughout their entire length by split lagging and boughs of trees. The snow on the mountains usually lasts until the end of April, and from then to the end of November rains are depended upon to supply the requisite amount of water.

At the Ready Bullion mine no provision has been made for a supply of fresh water for mill-use other than that for the boilers. Here salt water is pumped from the channel and used for all purposes in the mill. Experience has shown that while it is very destructive on all exposed iron in the batteries and on the vanners, it is better than fresh water for amalgamating purposes, but this advantage by no means compensates for the loss caused by its corrosive action on all exposed iron and on the vanner-belts. The coal—about 22,000 tons per year—is obtained from the mines on Vancouver island at reasonable rates. This coal is transported to the mines by means of barges, which on their return-trips carry concentrates in bulk to the Tacoma Smelter.

*Methods of Catching the Free Gold.*—The free gold is caught both inside and outside of the mortars by means of quicksilver. There is a diversity of opinion among the various amalgamators as to where, when, and in what quantities the quicksilver should be fed. Table IX. gives quantities used. The result of a series of tests in the various mills shows that the quantity of quicksilver fed in the mortars and on the plates varies directly with the gauge of the screen, and, consequently, with the coarseness of the ore. The coarser the crushing, the more quicksilver it is necessary to add to the mortars to make any saving at all.

TABLE IX.—Detail of Clean-Up and the Effect of Coarse Crushing on the Distribution of Amalgam.  
Ten Months, 1901-2.

Mill.	Tons Crushed Per Stamp in 24 Hours.	Oz. Quicksilver Fed Per Ton Ore Crushed.		Distribution of Clean-Up. Per Cent.							Value Amalgam Per Ounce.	Value Bullion Per Ounce.	Value Free Gold Per Ounce Quicksilver Fed.	Per Cent of Clean-Up in Free Gold.	Per Cent of Concentrates.	Value Tailings Per Ton.	Gross Value Per Ton Ore Crushed.
		Battery.	Total.	Blocks.	Barrel.	Vanners.	Tanks.	Traps.	Tail-Boxes.	Ashes.	Tables.						
Treadwell "240"	4.00	.2098	.0319	.2417	.0742	.1038	.0181	.0188	.0199	.0151	.0017	.6584	\$18.65	\$5.00	57.87	1.75	\$1.943
Treadwell "300"	4.59	.1264	.0695	.0896	.0961	.0282	.0107	.0136	.0128	.0015	.7475	18.66	18.70	4.15	67.85	1.65	.071
Mexican	4.33	.1794	.0845	.2639	.1952	.0793	.0082	.0284	.0362	.0087	.6550	6.48	18.70	4.15	67.85	1.65	1.842
700-Foot	3.97	.....	.....	.2022	.1893	.1285	.0111	.0167	.0262	.....	.6282	6.19	18.50	5.73	47.40	1.77	1.498
Ready Bullion	4.24	.1316	.0861	.2177	.8955	Blocks. } Barrel. } Table.		.0823	.0207	.0015	.....	5.94	18.25	6.12	70.00	1.74	2.058
Average	4.27	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.076	.....

Results from Ten Months, 1902-3.																	
Treadwell "240"	4.86	.0849	.0857	.1206	.....	.0901	.0216	.0119	.0303	.0277	.0011	.8172	\$6.12	\$18.65	45.99	2.00	\$1.104
Treadwell "300"	5.53	.0117	.0582	.0649	.....	.1067	.0483	.0357	.0366	.....	.0003	.7724	6.12	18.65	48.87	1.71	.092
Mexican	5.73	.2076	.0866	.2442	.0221	.0556	.0111	.0423	.0401	.....	.0031	.8257	6.48	18.70	48.87	1.83	1.979
700-Foot	5.70	.2299	.0279	.2578	.....	.0421	.0164	.0285	.0382	.....	.0019	.8788	6.19	18.50	48.36	1.83	1.933
Ready Bullion	5.34	.0824	.0381	.1205	.0012	.0963	.0069	.1936	.0444	.....	.....	.6557	5.94	18.25	53.75	2.07	.091
Average	5.38	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	48.04	.....	.094

On the other hand, the scouring action of the coarse sands on the plates necessitates frequent dressing to keep them well coated with quicksilver. It was the practice up to 1901 to keep the plates very wet, and even though the crushing was much finer an excessive amount of quicksilver was used. It will be seen from Table IX. that, although the tonnage crushed per stamp has shown a marked increase, the quantity of quicksilver used per ton is only about one-half the quantity formerly used.

The only amalgamated copper-plates used inside the mortars are the chuck-blocks. Two sizes are used at present (the 4 in. and 6 in. in height), but very little amalgam is collected from them. Formerly, they furnished 13.7 per cent of the amalgam collected, but since the fine screens were replaced they collect practically no amalgam, except during short periods in the winter when salt water is used. At these times the chuck-blocks become coated, but as soon as fresh water is again used the amalgam is scoured off, leaving the copper bare.

*Screens.*—The diagonal slot-screens which are used in all the mills are made of No. 23-gauge heavy Russian iron. Both the No. 4 and No. 5 are in use, these being equivalent to the 20- and 18-mesh wire screens, and are mounted in frames in the usual manner. Two widths are used, viz.: 9 in. and 12 in.—the former giving the better satisfaction.

A screen lasts about seven weeks in all the mills except the Ready Bullion, where it lasts only fifteen days. Salt water is used in this mill, and the corroding action seems to be intensified by the scouring of the sands in the mortar, which keeps the inside surface of the screen bright, thus always furnishing a fresh surface for the action of the water. These screens do not wear out as in the other mills, but become brittle and break.

Experiments are now in process with a make of iron-wire screen, but they have not advanced far enough for any conclusion to be reached.

*Apron-Plates.*—The motion of the battery-water, caused by the falling of the stamps, throws the pulp against the screen, and all particles fine enough pass through it and fall on the lip of the mortar. At the edge of the lip are placed two dashboards arranged in steps to stop the rush of the water and sand, and cause it to drop in a steady flow on the apron-plates,

which are placed immediately in front of the lip of the mortar, with the upper edge under the lip. This diminishes the scouring action, allowing the amalgam to collect near the upper end of the plate. The amalgam here is kept harder than at the lower end. This allows the lower end of the plate to be kept quite wet, which gives a better chance to catch the finer particles of gold that otherwise might float off.

These plates are made of the best Lake copper,  $\frac{3}{16}$  in. thick, 4 ft. wide, 10 ft. long, and are given a fall of 1.5 in. to the foot. The plates are prepared in the usual manner by cleaning with a weak solution of potassium cyanide and rubbing in quicksilver until the upper surface is thoroughly amalgamated. While in use they are dressed with quicksilver twice a day, and the time taken for dressing should not exceed four minutes per day.

At the lower end of each plate is placed a wooden trough lined with copper, called the tail-box, where very little amalgam is caught. From the tail-boxes the pulp is conveyed through 3-in. pipes to the mercury-traps. These traps are made of cast-iron in the shape of a four-sided truncated pyramid, having the smaller end down. The trap is 14 in. square at the top, 15.5 in. deep, with the lower end 6 in. square. In the bottom is a 2-in. tap closed by a plug for drawing off the contents when cleaning up. Inside the trap is a block 14 by 8 in. on top, 11 in. deep and 8 by 8 in. on the bottom. The pulp from the table enters the trap through a 3-in. pipe that reaches to within 2 in. of the bottom. It then flows up through the space between the trap and the wooden block, and thence over the block into the discharge-launders.

These boxes are usually suspended under the battery-floor and from them the pulp flows through launders, where it is divided and conveyed by 3-in. iron pipes to the distributing-boxes of two vanners.

On the floor of the distributing-box of each vanner is placed an amalgamated copper-plate, varying with the size of the vanner used, those on the 4-ft. vanner being 18 in. by 3 ft. 4.75 in., and on the 6-ft. vanners 18 in. by 5 ft. 9 in. From this plate the pulp flows over the vanner, the heavy particles, including sulphurets and some free gold, being saved, while the lighter pass over the tail of the vanner into the tailings-launders, which discharge into Gastineaux channel.

*Concentration.*—By reference to Table IX., it will be seen that 48 per cent of the value contained in the ore is recovered by concentration. For purposes of concentration two sizes of Frue vanners are used: the 4 ft. and 6 ft. These vanners are so arranged that the pulp from 5 stamps is divided between 2 vanners. This style of concentrator is very well adapted to the ore, and from a study of the tables given above it would be hard, indeed, to realize a much better saving. The wear and tear on the machines is very light in all the mills, with the exception of the Ready Bullion, where the vanners get more than their share of the destructive effects of salt water in use.

When a stamp is crushing 5.6 tons of ore in 24 hours, each stamp requires 4.25 gallons, and each vanner 1.5 gallons, of water per minute. In the mills where the 4-ft. vanners are used they are overloaded. This accounts for both sizes of vanners using the same amount of water.

There is a little less than 2 per cent of concentrates in the ore. The concentrated product has a value of about \$51 per ton in all the mills, with the exception of the Ready Bullion, where the concentrates assay about \$35 per ton.

In connection with each mill is a storage-bin for concentrates, holding about 400 tons. These bins are situated near the mill, and when the concentrates have been collected at the vanners and shoveled into cars, they are trammed to a small hydraulic elevator which raises the loaded car to the level of the top of the bin, where it is dumped. From these storage-bins the concentrates are drawn off through chutes into special cars holding 2.5 tons, and hauled by locomotives to the wharf, where they are dumped through chutes directly into the hatches of the barges which transport the concentrates to the Tacoma Smelter, where it is treated.

Table VIII., on p. 365, shows in detail the cost of the different departments of milling as well as the equipment and running time of the various mills for the last 10 months.

*Clean-Up.*—The clean-ups in the various mills are all conducted in the same fashion, and are so regulated that they will be finished by the 15th of each month.

The first day of the clean-up is devoted to the amalgam-traps and the tank in the amalgamating-room.

To clean a trap, 5 stamps are hung up and the feed-water shut off. When the pulp has ceased to flow through the trap, the wedge that holds the wooden center-piece is loosened and the center-piece removed, first being carefully washed to cleanse it of any adhering amalgam. Then the tap in the lower end is opened and the contents of the trap allowed to flow out into a small launder, which conveys the material to a central tank. The trap is then carefully washed out, the tap and wooden center-piece replaced, and it is ready for use. It takes an average of 5 minutes to clean each trap. When all the traps have been cleaned, the contents of the receiving-tanks after being roughly washed is collected and taken to the amalgamating-barrel for further treatment. In the meantime the contents of the tank in the amalgamating-room has been removed. This is added to the product from the mercury-traps, and the total charged into the amalgamating-barrel.

This barrel is made of cast-iron, 20 in. in diameter and 4 ft. long. It is supported in a horizontal position by iron trunnions cast in the head, and is driven at the rate of 15 rev. per min. by a belt leading to one of the vanner-countershafts.

The barrel is charged through a hand-hole in the top, which can be hermetically sealed; from 300 to 500 lb. of ore constituting a charge. From 75 to 125 oz. of mercury is then added, 6 iron cannon-balls put in to act as grinders, and the barrel filled with water. The hand-hole cover is then put on, and the barrel started revolving. The charge is left in the rotating-barrel 12 hours; the barrel is then opened and the charge allowed to run out into the amalgamating-pan. This pan is made of cast-iron 4 ft. in diameter. Around the edge, with the exception of a space 8 in. wide to serve as an outlet, is a rim 2.5 in. high. The bottom is made slightly concave to resemble a Mexican *batea*. The concentrating motion of the *batea* is imitated as closely as possible. This is obtained by means of an eccentric belt, driven from one of the counter-shafts of the mill. When the barrel is stopped, the cannon-balls are taken out and put in the pan, which is immediately started. By the motion of the pan the heavy contents are concentrated in the middle, while the lighter are washed off by means of a stream of water flowing through it, the concentrated product being kept in motion, and at the same time ground, by means of the cannon-balls.



When the concentrate is cleaned of all light material, the pan is stopped and the pieces of iron, etc., removed. The amalgam is then put in a pan, the finer particles of iron removed by means of a magnet, and the other foreign material by a sponge or other means. When the amalgam is clean, it is put in small cloth bags and the quicksilver pressed out by means of a hydraulic ram, designed by one of the mill-foremen. The pressed cakes of amalgam are weighed and sent to the assay-office to be retorted and melted into bullion.

The second and succeeding days are devoted to the cleaning of the batteries and amalgamating-plates. These are cleaned at the rate of 4 batteries of 5 stamps per day. (In the Treadwell 300-stamp mill, 5 batteries are cleaned per day.)

To clean up a battery the feed is shut off and the stamps allowed to drop until they begin to pound on iron, then they are hung up. The water is then shut off, and the splash-boards, curtains, screens and chuck-blocks are removed. The water remaining in the mortar is dipped out, and the coarse sand around the top of the dies shoveled into buckets to be put back into the mortar when the clean-up is over. The lip of the mortar and the plates are then carefully hosed off (a trough being first put in the tail-box to catch any loose amalgam) and the entire surface of the plate covered by a wooden cover for steaming. A space of 0.75 in. is left between the cover and the plate by means of three slats 0.75 in. thick nailed to the bottom of the cover. Sacks or other coverings are placed over the ends and edges to prevent the escape of steam; the end of a steam-hose is then introduced through a hole in the cover, the steam turned on and allowed to remain so from 20 to 30 minutes. In the meantime a second battery is prepared, and any renewals made ready, so that no time will be lost when the mortar is cleaned out. While the plate is being steamed the chuck-block is cleaned of any adhering amalgam, recoated with quicksilver and is ready to be replaced; while the sand-distributing box on the vanners, corresponding to the batteries shut down, are taken off and the amalgam removed from the copper-plate by means of chisels. This amalgam is collected, the plates dressed in the usual manner, the distributing-box replaced, and the vanner is ready for starting. When the plate has been sufficiently steamed, the steam is shut off and

the cover removed and taken to the next plate that has already been prepared for its reception. The steamed plate is then allowed to cool for a few seconds, when the operation of removing the amalgam commences. This is done by scraping the plates with sharp chisels, and as much amalgam as possible is removed without exposing the copper. The amalgam is then collected, taken to the amalgamating-room, and locked up for further treatment.

Two men now begin work on the mortar, and to protect the plates a wooden platform is placed at the head for the men to stand on. If there are no renewals necessary (but this is unusual) only a portion of the sand is taken out. If necessary, the shoes are removed by driving a wedge through the eye left in the boss-head just above the end of the shank of the shoe and forcing it out. The sand is then dug out of the mortar by means of sharp-pointed hand-picks and scoops, the die and liners removed, and the mortar thoroughly cleaned. All pieces of iron, together with the worn-out shoes and dies and liners, are taken to the amalgamating-room to be thoroughly cleaned, and the heavy sand taken to the clean-up barrel.

The liners and false bottoms are then put in and the die set on the false bottom, while the fine sand first removed from the mortar is tamped around the die to hold it in place. The shoe is then set on the top of the die with a collar of wooden shims around its neck. A 3.25-in. block is then placed on the top of the neck of the shoe, and the stamp lowered until the boss-head rests on the block. The keys of the tappets are loosened and the tappet is allowed to fall down to the finger, where the keys are tightened. If the shoe has not been removed, a 9.25-in. block is placed on top of the die and the tappets set as above. Each stamp is then successively dropped and hung up, when the shoe is firmly fixed in the boss-head. The recesses for the chuck-block, screens, etc., are washed out, and the chuck-block, screens and dash-boards put in place. The plate is then washed with a weak solution of cyanide, when quicksilver is sprinkled over its surface and thoroughly rubbed in with whisk-brooms. The quicksilver is evenly distributed by rubbing with cloths moistened with a weak solution of cyanide. Some ore is now fed into the mortar, working the feeder by hand, the water turned on, the small clean-up trough removed from the tail-

box, and the stamps allowed to drop. Particular care is paid to the feeding of ore when the stamps are started, as the mortar is empty of all ground material.

The heavy sands from the mortar are treated in the clean-up barrel in the manner described above, while the amalgam removed from the plates and chuck-blocks is simply ground in the clean-up pan and the amalgam cleaned in the usual manner.

*Labor.*—Table X. gives the number, length of shifts and wages of all men working in and about the mills. During the summer, and often in the winter, months the mills are run entirely by water, and at such times the engineers, firemen and coal-passers are not required. When steam is used, the wages of firemen and coal-passers, and at the Ready Bullion that of the engineers, is divided between the mine and mill in proportion to the horse-power used by each.

Each mill is in charge of a foreman who is responsible for its condition and efficiency. He also attends to the handling and cleaning of all amalgam collected.

The duty of the amalgamators is to dress and keep the plates and chuck-blocks in good condition, to set tappets, regulate water-supply and make all renewals; or, in other words, to keep the crushing-department of the mill up to its maximum efficiency.

The feeders attend to the uniform feeding of the ore to the batteries, and assist the amalgamators in making renewals and during the clean-up. A good feeder is as valuable a man as there is in a mill.

The vanner-man attends exclusively to the running and feeding of the vanners. It is requisite that these men have considerable experience; in fact, an order has just been issued making it necessary for a man to serve first in the capacity of sulphide-puller before being put in charge of vanners.

The titles of the other men about the mills explain their duties.

*Costs.*—The details of the receipts and expenditures per ton of ore treated at the mills of the Treadwell group of mines, as well as the conditions affecting them, are given in extenso in the subjoined tables; Table XI. referring to the Alaska Treadwell Gold Mining Co., Table XII. to the Alaska Mexican Gold Mining Co. and Table XIII. to the Alaska United Gold Mining Co., the last-named company operating the Ready Bullion mine and the "700-Foot" claim.

TABLE X.—*Mill Labor.*

Title.	Treadwell Mills.				Mexican.			Ready Bullion.			700-Foot.		
	300-MILL.		240-MILL.		Number of Men.	Length of Shift.	Rate of Wages.	Number of Men.	Length of Shift.	Rate of Wages.	Number of Men.	Length of Shift.	Rate of Wages.
	No. of	Rate of Wages.	No. of	Rate of Wages.									
	Hrs.	of Shift.	Hrs.	of Shift.									
Foreman .....	1	\$150.00	1	\$150.00	1	12	\$150.00	1	12	\$150.00	.....	.....	.....
Amalgamators .....	4	90.00	4	90.00	2	12	90.00	2	12	90.00	2	12	\$90.00
Feeders .....	8	70.00	8	70.00	4	12	70.00	4	12	70.00	4	12	70.00
Vannermen .....	4	65.00	4	65.00	2	12	65.00	2	12	65.00	2	12	65.00
Oilers .....	2	65.00	2	65.00	2	12	65.00	2	12	65.00	2	12	65.00
Sulphide-pullers..	2	2.00	2	2.00	1	10	2.00	1	10	2.00	1	10	2.00
Sulph.-shovelers..	2	2.00	2	2.00	1	10	2.00	1	10	2.00	1	10	2.00
Engineers .....	.....	.....	.....	.....	2	12	2.50	2	12	2.50	.....	.....	.....
Firemen .....	.....	2.50	2	2.50	2	12	2.50	2	12	2.50	.....	.....	.....
Coal-passers .....	.....	2.00	2	2.00	2	10	2.00	2	10	2.00	.....	.....	.....
Crusher-men .....	4	2.25	4	2.25	2	10	2.25	2	10	2.25	2	10	2.25
Repairs.													
Vanners .....	1	100.00	1	100.00	.....	.....	.....	1	12	3.00	.....	.....	.....
Carpenters .....	1	4.00	.....	4.00	.....	10	4.00	.....	.....	.....	.....	.....	.....
Laborers .....	1	2.00	.....	2.00	.....	10	.....	.....	10	2.00	.....	.....	.....
Total .....	30	.....	34½	.....	21½	.....	.....	22½	.....	.....	14	.....	.....

NOTE.—The above wages include board and lodging. Engineers, firemen and coal-passers are only employed part of the time by the mills. (a) Amounts of \$65 or greater refer to monthly wages, and those of \$4 or less to daily wages.

TABLE XI.—*Alaska Treadwell Gold-Mining Co.*

Receipts and Expenditures in Dollars Per Ton. Also Conditions Affecting Operating Costs.

Year.	Total Gross Bullion and Concentrate Yield Per Ton.		All Expenses Per Ton.				Profits Per Ton.		Per Cent. Ore From.		No. Feet.		Pounds Powder Used		Per Cent. Ore Milled By.		Per Cent. Indian Labor.		Monthly Expenses Per Ton During Year.				Grand Total.		Remarks.						
			Douglas Island.		San Francisco, London, Paris, and Consulting Engineer.	Total.	Operating.		Other.	Total.	Surface-Pits.	Underground.	Shaft Sunk.	Other Dev. Work.	Pounds Powder Per Ton Ore.	Water-Power.	Steam-Power.	Per Cent. Indian Labor.	Average Daily Pay All D. Island Employees.	Maxi-mum.		Mini-mum.		Expenses.		Value Per Ton.	Tons Ore Milled.	Profit.	Year.		
																				Supplies.	Labor.	Month.	Expenses.							Month.	Expenses.
1890-1.....	\$3.49		0.73	0.85	0.13	1.71	1.71	0.10	1.80	100					1.01	79.11	20.89							2.66	\$10.65	220.686	\$118,208.90	1890-1			
1891-2.....	2.95		0.65	0.78	0.07	1.50	1.45	0.08	1.51	100					0.93	73.29	26.71								2.10	32.08	239.633	361,980.16	1891-2		
1892-3.....	2.94		0.72	0.56	0.07	1.35	1.59	0.04	1.63	100					0.76	70.19	29.82								1.80	41.42	237.235	835,613.79	1892-3		
1893-4.....	3.21		0.52	0.75	0.09	1.35	1.86	0.10	1.96	100					0.73	64.37	35.63	21.34	3.00						1.84	46.45	220.013	429,948.86	1893-4		
1894-5.....	2.60		0.59	0.71	0.07	1.37	1.23	0.06	1.23	100					0.69	71.34	28.66	15.84	3.15						1.73	50.52	241.278	399,531.56	1894-5		
1895-6.....	2.97		0.45	0.65	0.07	1.16	1.81	0.07	1.88	99					0.76	80.56	19.44	13.63	3.10						1.65	57.73	263.670	497,342.22	1895-6		
1896-7.....	2.80		0.75	0.75	0.07	1.57	1.23	0.11	1.34	95.1					0.97	74.66	25.34	14.47	3.25						1.63	48.14	242.027	323,031.95	1896-7		
1897-8.....	2.32		0.57	0.82	0.08	1.48	0.84	0.12	0.96	89.5					0.84	62.73	37.27	8.36	3.23						1.60	44.33	254.329	243,200.82	1897-8		
1898-9.....	2.71		0.48	0.74	0.07	1.28	1.43	0.12	1.55	85.1					0.88	68.46	31.54	12.13	3.13						1.63	48.11	250.408	386,792.34	1898-9		
1899-0.....	2.07		0.44	0.46	0.03	0.92	1.15	0.06	1.21	67.0					0.80	79.97	20.03	5.71	3.02						1.73	38.78	557.960	673,961.18	1899-0		
1900-01.....	1.88		0.54	0.62	0.03	1.19	0.69	0.08	0.77	47.7					0.76	77.39	22.61	5.33	3.25						1.65	41.92	457.802	852,558.88	1900-01		
1901-02.....	1.91		0.49	0.78	0.01	1.23	0.63	0.05	0.68	56.0					0.82	77.51	22.61	5.00	3.26						1.59	40.81	462.893	463,489.93	1901-02		
1902.....	1.88		0.46	0.54	0.01	1.01	0.82	0.05	0.87	53.3					1.34	84.33	15.07	4.8	3.18						1.99	49.79	508.636	442,513.32	1902.....	Six months.	



TABLE XIII.—*Alaska United Gold-Mining Co. Ready Bullion Mine.*

Receipts and Expenditures in Dollars Per Ton. Also Conditions Affecting Operating Costs.

Year Ending Dec. 31st.	All Expenses Per Ton.										Profits Per Ton.			Per Cent Ore From.			No. Feet.	Pounds Power Used			Per Cent. Ore Milled By.		Monthly Expenses Per Ton During Year.				Concen- trates.		Grand Total.	Year Ending Dec 15th.	Remarks.
	Douglas Island.		San Francisco, London, Paris, and Consulting Engineer.		Total.	Operating.	Other.	Total.	Surface-Pits.	Underground.	Shaft Sunk.	Other Dev. Work.	Pounds Power	Water-Power.	Steam-Power.	Per Cent. Indian Labor.		Average Daily Pay All D. Island Employees.	Maxi- mum.	Month.	Expenses.	Min- imum.	Month.	Expenses.	Per Cent.	Value Per Ton.	Tons Ore Milled.	Profit.			
	Supplies.	Labor.	Total.	Other.																											
					Total.	Operating.	Other.	Total.	Surface-Pits.	Underground.	Shaft Sunk.	Other Dev. Work.	Pounds Power	Water-Power.	Steam-Power.	Per Cent. Indian Labor.		Average Daily Pay All D. Island Employees.	Maxi- mum.	Month.	Expenses.	Min- imum.	Month.	Expenses.	Per Cent.	Value Per Ton.	Tons Ore Milled.	Profit.			
Total Gross Bullion and Concentrate Yield Per Ton.	Operating.	Other.	Total.	Surface-Pits.													Underground.												Shaft Sunk.	Other Dev. Work.	Pounds Power
1898.....	\$2.42	0.71	0.55	0.03	1.29	1.13	0.07	1.19	100	00	602	2112	2.69	.....	100	.....	8.84	.....	.....	.....	.....	.....	.....	2.03	\$72.00	19,612	\$23,412 13	1898			
1899.....	2.72	0.90	1.04	0.03	1.97	0.75	.....	0.75	87.98	62.02	47	4272	1.82	.....	100	9 233.08	2.62	Mar.	1.62	1.92	38	10	102,107	121,339	75,1899	121,339 75	1899				
1900.....	2.15	0.78	0.83	0.03	1.67	0.48	.....	0.48	8.42	91.58	259	3552	1.19	.....	100	8,503.09	2.84	Oct.	1.40	2.07	31	57	179,410	86,418	18,1900	86,418 18	1900				
1901.....	1.94	0.88	0.90	0.03	1.86	0.08	.....	0.08	4.17	95.83	165	2370	1.28	.....	100	2,333.05	3.48	Mar.	1.13	1.97	33	14	170,642	14,476	61,1901	14,476 61	1901				
1902.....	1.48	0.70	0.78	0.02	1.50	0.02	.....	0.02	1.43	98.57	58	2787	1.19	.....	100	1,003.10	.....	.....	.....	.....	.....	.....	2.04	32.95	226,522	(Loss) 5,865.70	1902				
700-Foot Claim.																															
1899.....	\$1.72	0.54	1.10	0.03	1.63	0.04	.....	0.64	73.02	25.98	.....	1621	1.11	100	.....	1,873.04	3.23	May	1.01	1.94	37	87	85,065	3,405.88	1899						
1900.....	1.81	0.69	0.84	0.03	1.56	0.25	.....	0.25	49.89	50.11	72	485	1.94	58.42	58.3	373.01	2.31	Mar.	1.18	1.73	31	91	125,612	31,571.08	1900						
1901.....	1.50	0.90	1.05	0.03	1.97	0.48	.....	(Loss) 0.48	23.14	76.86	312	1396	1.03	51.16	45.84	1,323.11	4.10	July.	1.80	1.71	28	13	89,450	(Loss) 42,797.49	1901						
1902.....	1.33	0.52	0.63	0.02	1.17	0.16	.....	0.16	.....	100	.....	362	0.78	81.82	18.18	.....	3.10	.....	.....	.....	.....	.....	1.84	48.46	118,541	19,302.64	1902				

## The Reduction of Lead from Litharge in Preliminary Assays, and the Advantages of an Oxide Slag.

BY E. H. MILLER, E. J. HALL AND M. J. FALK, NEW YORK CITY.

(New York Meeting, October, 1903.)

It is well known that, with the exception of the so-called "niter-and-nails" method, all methods used for the fire-assay of sulphide-ores,\* especially those of iron, zinc, lead and copper, are very tedious and require the constant attention of the assayer. The objections to the niter-and-nails method are: The liability of the charge to boil over, due to the presence of the niter; the low results in silver (uncorrected); and the frequently large discrepancy between the actual weight of the lead button and the calculated weight. In many cases the buttons are too small to collect all of the gold and silver contained in the sample, while in others they are too large for cupellation. It was thought that, if these objections to the niter method could be eliminated, it would give a method for the crucible-assay of sulphide-ores, and especially the low-grade ores of zinc and copper, which would be superior to any method in use at the present time. The only similar methods of assay are those of Perkins,† Rhead and Sexton,‡ and Mitchell.§ The method of Perkins requires too much litharge for it to be a commercial success; that of Rhead and Sexton, utilizing the preliminary assay we found to be the best;—the charge for the final assay, however, was modified by having borax-glass added to it; also, the oxidizing power of the niter was determined by a charge which differs from either the preliminary or the

\* Mitchell, *Manual of Practical Assaying*, 1872, pp. 464–469. Roswag, *Métallurgie d'Argent*, pp. 112–113. Collins, *Metallurgy of Lead*, p. 15. Percy, *Metallurgy of Silver and Gold*, p. 245. Furman, *Manual of Practical Assaying*, 1898, pp. 126 et seq. J. A. Phillips, *Mining and Metallurgy of Gold and Silver*, pp. 505 et seq. Beringer, *Manual of Assaying*, pp. 95 et seq. S. Campredon, *Guide Pratique du Chimiste Métallurgiste et de l'Essayeur*, pp. 292 et seq. Ricketts and Miller, *Notes on Assaying*, pp. 92 et seq. Rhead and Sexton, *Assaying and Metallurgical Analysis*, pp. 101 et seq.

† Perkins, *Trans.*, xxxi., p. 913.

‡ Rhead and Sexton, *Assaying and Metallurgical Analysis*, p. 105.

§ Mitchell, *Manual of Practical Assaying*, 1872, pp. 464 et seq.



final assay,—and the method of Mitchell, using litharge only, requires too much litharge for fluxing the impurities to render it applicable in this case. The last-named method has long been abandoned.

### THE PRELIMINARY ASSAY.

In order to use any of the niter methods, the oxidizing power of the niter must first be accurately determined,\* and for this purpose the reducing powers of the reagents named in the following list were obtained: Argol, flour, charcoal, sulphur, sugar and iron sulphide ( $\text{FeS}_2$ ); later, the oxidizing power of niter was determined in connection with each of these reducing agents. The tests were made in crucibles manufactured by the Denver Fire-Clay Co., in order not to complicate the results, for the reason that the fusion of a basic charge in a Hessian crucible would cause the absorption of silica from the walls of the crucible itself. It was found afterward that Battersea clay-crucibles withstood the charge finally used better than the crucibles of the Denver Fire-Clay Co. The cover to the charge was of salt alone.

The charges used and the results are given in the following table:

TABLE I.—*Assay-Tests for Reducing and Oxidizing Power.*

		Quantity Used (in Grammes).					
		Argol.	Flour.	Sugar.	Char-coal.	$\text{FeS}_2$ (90 Per Cent).	Sul-phur.
Charge .....	Reducing-agent.	2.5	2.5	2.5	0.75	2.5	1.
	Litharge ( $\text{PbO}$ ).	45.	45.	45.	45.	45.	45.
	Sodium bicarbonate ( $\text{NaHCO}_3$ ).....	10.	10.	10.	10.	10.	10.
	Silica ( $\text{SiO}_2$ ).....	7.	7.	7.	7.	7.	7.
Lead button obtained (average).....		24.03	27.32	29.46	19.56	23.33	18.11
Lead button obtained after the addition to each charge of 3 grammes of niter.....		13.43	14.65	15.54	5.71	14.55	7.34
	(Quantity of lead oxidized by 1 gramme of niter).....	3.51	4.22	4.62	4.61	3.59	2.93

\* Ricketts and Miller, *Notes on Assaying*, p. 43.

It is an accepted fact that the oxidizing power of niter depends upon the acid or basic character of the charge;\* furthermore, as shown by the results of the experiments given above, the oxidizing power varies also according to the reducing agent with which it is used, even under exactly similar conditions of temperature and slag; a fact which was not anticipated, and the discovery of which throws much light upon the troubles encountered in preliminary assays.

In order to make a systematic study of this subject, it was deemed best to begin with the simplest combination, and for this purpose an ore was selected which was nearly pure pyrite. The sample was ground to 100-mesh size and the iron and sulphur contents were accurately determined as follows: Sulphur, 47.41 per cent. by the bromine method,† and 47.80 per cent. by the nitric and hydrochloric acid method‡ (average 47.6 per cent.); iron, 41.96 per cent. by the potassium permanganate titration method,§ which gives a total content of 89.56 per cent. of  $\text{FeS}_2$ , an amount which may be taken as 90 per cent. for all practical purposes.

#### VARYING THE CHARGES.

*Charge of Litharge, Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ) and Silica.*—For the purpose of varying the charges, 3 grammes of iron sulphide,  $\text{FeS}_2$  (90 per cent.), were mixed with 70 grammes of litharge, and to this mixture different proportions of silica were added, and each of the different charges was submitted to the fire-assay. The weights of the lead buttons obtained under these different conditions are given in the subjoined table:

TABLE II.—*Assay-Results Obtained by Adding Different Quantities of Silica to a Charge of 70 Grammes of Litharge and 3 Grammes of Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ).*

Quantity of Silica Added.	Grammes 0.	Grammes 5.	Grammes 10.	Grammes 15.
Weight of resultant lead .....	<div> <div>[</div> <div>20.15 21.50 21.00 .....</div> <div>]</div> </div>	<div> <div>11.83 11.25 10.50 .....</div> </div>	<div> <div>3.1 2.8 2.6 4.8</div> </div>	<div> <div>0.95 2.28 0.10 .....</div> <div>  </div> </div>

\* Beringer, *Manual of Assaying*, p. 98.

† Gladding, *Journal of the American Chemical Society*, 1894, p. 403.

‡ Cairns, *Quantitative Analysis*, p. 116.

§ *Idem.*, pp. 64, 116.

|| It was impossible to pour this charge, which accounts for the irregular results.

The slag obtained from the charge having no silica was black, brittle and opaque; it contained lead sulphide, clung to the lead button and could not be poured. The slags from the charges containing 5 and 10 grammes of silica, had similar characteristics, but could be poured. In all of these cases the most notable effect was the distribution of lumps of infusible material throughout the slag. The lead buttons contained sulphides and were hard, brittle and easily lost. The temperature at which the tests were made was  $975^{\circ}\text{C}$ ., as measured by a Le Chatelier thermo-electric pyrometer, and the time required for complete fusion was 25 minutes. The results given in Table II. show that the addition of silica diminishes the quantity of lead obtained by the fusion until a point is reached, at which scarcely any is reduced. In the reaction between iron sulphide and litharge, it is reasonable to expect that the sulphide-sulphur would be oxidized to sulphur tri-oxide, according to the equation



which would give a button of 34.04 grammes of lead for the pyrite used.

In the tests, however, this was not the case, as the weight of the lead obtained was not in accordance with the reaction. The discrepancies in the results are due to the fact that there was still sulphide-sulphur remaining in the slag. The addition of silica to the charge seemed to emphasize this condition, for, when 10 and 15 grammes of silica were added, the quantity of sulphide-sulphur remaining in the slag was considerable, while the proportion of the sulphur which actually reduced the lead was very small indeed. From these results it will be seen, that a method in which an acid-slag is used cannot be applied to the preliminary assay of sulphide-ores, due to the fact that it is impossible to control accurately the acidity of the charge in such a way as to compel the sulphur to oxidize to a certain degree,—that is, to  $\text{SO}_2$ . Furthermore, the lead reduced in acid-charges depends greatly on the temperature used. This partly explains the trouble of the old preliminary assay, in which the preliminary charge almost always differed from the charge used in the final assay.

*Charge of Litharge, Borax-Glass and Iron Sulphide* ( $\text{FeS}_2$ , 90 Per Cent.).—In order to determine the action of litharge and borax, charges were made of 3 grammes of iron sulphide ( $\text{FeS}_2$ , 90 per cent.), 70 grammes of litharge and different quantities of borax-glass. The results of the assays are given in the subjoined table:

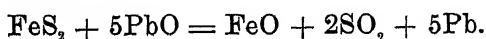
TABLE III.—*Assay-Results Obtained by Adding Different Quantities of Borax-Glass to a Charge of 70 Grammes Litharge and 3 Grammes of Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ).*

Quantity of Borax-Glass Added.	Grammes 5.	Grammes 10.	Grammes 15.	Grammes 20.
Weight of resultant lead.....	21 50	23.50	22.95	No results.*
	21.75	24.00	Others irregular	.....
	22.73	23.50	.....	.....

\* The lead produced did not collect in a button.

The slags formed in these tests were readily poured, easily fused, and resembled flint both in the brownish-black color and in the hardness; they adhered to the lead buttons, contained sulphur and yielded hydrogen sulphide upon the addition of hydrochloric acid. The lead buttons obtained were hard and brittle, and, in the charges containing more than 5 grammes of borax-glass, were separated with difficulty from the slag and were easily lost; they contained less sulphide but yielded hydrogen sulphide upon the addition of hydrochloric acid.

The temperature at which assays were made was  $975^\circ \text{C}$ ., and the time of fusion was 25 minutes. The effect of borax-glass in preventing the oxidation of the sulphur is not as marked as that of silica, yet it is sufficient to prevent the use of borax for any niter-method. The charge as a whole is very difficult to handle, as it yields a slag which is so hard that it requires considerable hammering to separate it from the button. The buttons themselves are very brittle and frequently remain "shot" throughout the slag, which makes it tedious to collect them. The results given in Table III. agree with the oxidation of the sulphur to sulphur dioxide ( $\text{SO}_2$ ) according to the following reaction:



That is, for 3 grammes of 90-per cent.  $\text{FeS}_2$ , 23.27 grammes Pb should be reduced. This calculated quantity agrees closely with the results obtained in Table III.

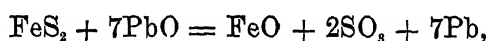
*Charge of Litharge, Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ), and Sodium Bicarbonate.*—In order to determine the action of litharge and sodium bicarbonate,  $\text{NaHCO}_3$ , charges were made up, using 3 grammes of pyrite, 70 grammes of litharge and different quantities of sodium bicarbonate. The effect of this charge was very curious. The resultant slags were in two layers, the upper one containing all the salt, soda and sulphur in the form of sodium sulphate ( $\text{Na}_2\text{SO}_4$ ), while the lower contained only  $\text{FeO}$ ,  $\text{Fe}_2\text{O}_3$  and  $\text{PbO}$ . The ratio of  $\text{FeO}$  to  $\text{Fe}_2\text{O}_3$  in the slag was found by analysis to be about 1 : 5. The results obtained with this charge are shown in Table IV.:

TABLE IV.—*Assay-Results Obtained by Adding Different Quantities of Sodium Bicarbonate to a Mixture of 70 Grammes of Litharge and 3 Grammes of Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ).*

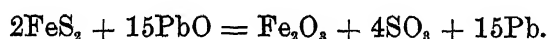
Quantity of Sodium Bicarbonate Added.	Grammes 5.	Grammes 10.	Grammes 15.	Grammes 20.
Weight of resultant lead.....	$\left\{ \begin{array}{l} 32.22 \\ 31.68 \\ 33.50 \end{array} \right.$	$\left\{ \begin{array}{l} 34.30 \\ 33.62 \\ 32.06 \end{array} \right.$	$\left\{ \begin{array}{l} 32.08 \\ 32.42 \\ 32.07 \end{array} \right.$	$\left\{ \begin{array}{l} 31.86 \\ 32.58 \\ 32.17 \end{array} \right.$

The slags from these charges were very fluid and could be poured as easily as water; they were readily fusible and, on being cooled, the upper layer puffed up. The lead buttons separated readily from the slag; they contained no sulphur, and were soft, malleable and of bright appearance, with the exception of those from the charges containing much sodium bicarbonate, which yielded a crumbly product. The temperature used for fusion was  $975^\circ \text{C}$ . and the time of each test was 20 minutes.

The reactions which occurred were as follows :



and



The theoretical weight of the lead button which would result from the first reaction is 32.58 grammes, while that from the second is 34.04 grammes, and as both  $\text{FeO}$  and  $\text{Fe}_2\text{O}_3$  were formed in the slag, the results obtained in the tests, as given in Table IV., agree very closely with the theoretical quantities.

*Charge of Lead Silicates and Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ).—*

In order to determine the effect of  $\text{FeS}_2$  on lead silicates, the latter were made up as follows, and then ground to 80-mesh size.

I.  $2\text{PbO} : 1\text{SiO}_2$  or 446 grammes  $\text{PbO}$  to 60 grammes  $\text{SiO}_2$ .

II.  $2\text{PbO} : 2\text{SiO}_2$  or 446 grammes  $\text{PbO}$  to 120 grammes  $\text{SiO}_2$ .

III.  $2\text{PbO} : 3\text{SiO}_2$  or 446 grammes  $\text{PbO}$  to 180 grammes  $\text{SiO}_2$ .

The results obtained by fusing 3 grammes of  $\text{FeS}_2$  with each of these silicates were as follows:

I. Gave no button below  $950^\circ \text{C}$ .; above this temperature there was a partial reduction.

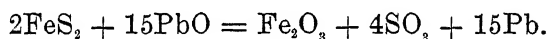
II. and III. gave a button of lead sulphide above  $950^\circ \text{C}$ ., but the charge was not fused below  $950^\circ \text{C}$ .

These results help to verify those shown in Table II., in which the quantity of lead reduced decreases with the increase of silica in the charge. Here, no button at all was obtained, except a button of lead sulphide in experiments II. and III., and a button of lead in I. above  $950^\circ \text{C}$ ., owing to the fact that the silica is not in excess. This explains why the buttons obtained in the tests given in Table II. were small; lead silicates were formed in the charge and the pyrite could not reduce any lead from them; furthermore, it proves the inadvisability of working with an acid-slag, and in charges acid with silica, it explains why the buttons come down full of lead sulphide and are therefore brittle. The preceding work shows that for sulphide-ores, the soda-litharge charge, used in Table IV., was the best. For this reason the following charges were fused, in which the litharge was increased from 70 to 100 grammes, in order to ascertain if the proportion of litharge present would have any effect. The results are given in the following table:

TABLE V.—*Assay-Results Obtained by Increasing from 70 Grammes to 100 Grammes the Quantity of Litharge Added.*

Quantity of Sodium Bicarbonate Added.	Grammes 5.	Grammes 10.	Grammes 15.	Grammes 20.	Grammes 25.
Weight of resultant lead button.....	{ 33.61 33.54 34.25 .....	{ 34.66 35.31 33.30 .....	{ 34.15 34.83 35.18 .....	{ 33.90 34.85 34.15 34.95	{ ..... ..... 34.13 .....

The slags obtained in these tests consisted of layers of double oxide, salt, sodium carbonate and sodium sulphate; they fused easily, poured readily, and did not adhere to the crucible. The buttons obtained were soft, malleable and clean; those with the small quantities of sodium bicarbonate being better than those with larger quantities. When much sodium bicarbonate was used (25 grammes) the buttons were scarcely malleable and did not collect well. The slight increase in the weight of the buttons, resulting from the charges having large quantities of sodium bicarbonate, was due to the oxides included in the button and adhering to the surface. The results are in accordance with the following reaction:



In order to ascertain if smaller quantities of litharge would be sufficient for the preliminary assay, charges were made up, using 10 grammes of sodium bicarbonate and quantities of litharge varying from 50 to 100 grammes. The following results were obtained:

TABLE VI.—*Assay-Results Obtained by Adding to 10 Grammes of Sodium Bicarbonate and 3 Grammes of Iron Sulphide (90 Per Cent.  $\text{FeS}_2$ ) Quantities of Litharge, Varying from 50 to 100 Grammes.*

Quantity of Litharge Added.	Grammes 50.	Grammes 70.	Grammes 100.
Weight of resultant lead button..	{ 33.61 33.54 34.25	{ 34.30 33.62 32.06	{ 34.66 35.31 33.30

The slags and the buttons obtained were similar to those obtained with previous tests. Table VI. shows that, for the preliminary assay, 50 grammes of litharge were sufficient, even though the litharge ought to be increased to from 70 to 100 grammes for the final assay. By this means the cost of the method is diminished. At this point it was thought necessary to redetermine the oxidizing power of the niter when used with the soda-litharge charge. The results obtained are given in Table VII.:

TABLE VII.—*Assay-Tests for the Determination of Oxidizing Power of Niter.*

Other Component of the Charge.	Quantity of Lead Oxidized by 1 Gramme of Niter.
	Grammes.
Iron sulphide (90 per cent. $\text{FeS}_2$ ).	4.73
Sulphur.....	4.40
Charcoal.....	5.15
Flour .....	5.09
Argol.....	4.76

These results show the absolute necessity of determining the oxidizing power of the niter with that substance or compound and charge with which it is subsequently to be used. The reducing power of different substances when used with the soda-litharge charge were also determined, the results being as follows:

TABLE VIII.—*Assay-Tests for the Determination of Reducing Powers.*

Other Component of the Charge.	Quantity of Lead Reduced by 1 Gramme of Reducing-Agent.
	Grammes.
Argol.....	9.61
Flour .....	10.53
Sugar .....	11.78
Charcoal.....	26.45
Sulphur.....	18.11
$\text{FeS}_2$ .....	12.22 (calculated)
$\text{FeS}$ .....	8.71
$\text{Sb}_2\text{S}_3$ .....	7.17
$\text{Cu}_2\text{S}$ .....	4.38
$\text{ZnS}$ .....	8.16

These figures are nearer the theoretical quantities calculated for the sulphide-ores than are those usually given in text-



books on assaying. This soda-litharge slag is similar to that used by Perkins,\* except that no silica is used and a much smaller quantity of litharge (70 grammes), as compared with 8 assay tons for  $\frac{1}{10}$  assay ton of matte, or 5 assay tons for 0.5 assay ton of ore.

Tests were made to obtain with the niter, in the final assay, a lead button of about 20 grammes, as calculated from a preliminary assay using the soda-litharge charge. For this purpose, charges were made up as follows:

<i>Preliminary Assay.</i>	<i>Final Assay.</i>
Ore (90 per cent. $\text{FeS}_2$ ), 3 grammes.	Ore (90 per cent. $\text{FeS}_2$ ), 0.5 assay tons.
Litharge, 50 grammes.	Litharge, 70 grammes.
Soda, 10 grammes.	Soda, 15 grammes.
	Niter, as calculated for a 20-gramme button.
Salt cover.	Salt cover.

The weight of the buttons obtained in the final assay agreed so closely with the 20-gramme buttons calculated, that this method proved to be more accurate than any other method heretofore used. (See Table IV.)

The first charge used in any niter method for sulphide-ore† was made up with litharge and niter only. This was abandoned later as inaccurate, for the reason that the sulphur was not completely oxidized, some even remaining as sulphide-sulphur in the slag.

The next charge tried, based apparently on the idea that acid charges seemed to work better than basic ones, was acid with silica, the fluxes being litharge and soda. This, again, was given up as inaccurate. As is shown in Table II., the more acid the charge, the less lead is reduced; therefore, the acidity of the charge had to be very accurately controlled, and as this was impossible, due partly to the gangue of the ore, the method was given up as inaccurate, but for no explained reason.

The third method,‡ which was the one in general use up to the present time, contained soda and litharge as fluxes, with the addition of from 5 to 20 grammes of borax-glass. This

\* Perkins, *Trans.*, xxxi., p. 913.

† Mitchell, *Manual of Practical Assaying*, pp. 464 et seq.

‡ Rhead and Sexton, *Assaying and Metallurgical Analysis*, p. 105.

method worked very well for ores low in sulphur, but when applied to high-grade sulphurets the results were very poor.\*

Borax glass has exactly the same effect as silica, although to a lesser degree (see Table III. *et seq.*); it also gives a slag and button containing sulphide-sulphur, showing that with borax-glass the oxidation is incomplete, and therefore unreliable.

The method described above, using soda and litharge, seems to be accurate. The sulphur is completely oxidized, even though the charge be varied slightly. The variation in the results is confined to the extent to which the iron is oxidized. This seems to be dependent on the excess of litharge, and does not materially affect the assay-results. As this method appears to be constant and to follow a definite reaction, we have applied it to ores, as described in the following section.

#### TESTING THE NEW CHARGE WITH ORES.

Ores of different composition, and having a reducing power due to sulphur, were assayed with the following soda-litharge charges:

<i>Preliminary Assay.</i>	<i>Final Assay.</i>
Ore, 3 grammes.	Ore, 0.5 assay tons.
Litharge, 50 grammes.	Litharge, 70 grammes.
Soda, 10 grammes.	Soda, 15 grammes.
	Niter (calculated for 20-gramme buttons).

The table on page 398 shows results obtained by the new method, as compared with those obtained by roasting, niter-nails, or scorification methods:

The slags and buttons in all cases had the characteristics of those of the typical soda-litharge charges. The most curious fact to be noted was that the charge does not boil over as was often the case with the old niter methods. Quick, hot fires (above 900° C.) and short fusions (from 15 to 20 min.) work best. The tests were made in a muffle, and the assays were run without covers, as it was found that the salt cracked the crucibles. The Battersea clay-crucibles withstood eight charges each, while those of the Denver Fire-Clay Co. would withstand about three charges. The best size of Battersea crucible to use is the 10-gramme size for the preliminary assay and either the 10- or the 20-gramme size for the final assay, depending on the size of the charge.

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\* Ricketts and Miller, *Notes on Assaying*, p. 94, note.

TABLE IX.—*A Comparison of the Results Obtained by the New Method with those of the Roasting, the Niter-and-Nails and the Scorification Methods.*

Number of Ore.	Lead from Preliminary Assay in Grammes.	Niter in Grammes Necessary for 24-Gr. Button on $\frac{1}{2}$ A. T. Ore-Charge.	Lead Button Obtained.	Silver in Oz. Per Ton. New Method.	Silver in Oz. Per Ton. Old Methods.	Gold in Oz. Per Ton. New Method.	Gold in Oz. Per Ton. Old Methods.	Remarks.
21	16.01	12.4	18.71	1.46	1.38*	0.86	.86*	All contain $\text{FeS}_2$ (basic or acid gangues). Ore No. 160 contains 9 per cent. Cu.
	15.53		18.35					
32	2.07	0.9‡	17.75	5.23	5.18*	0.38	.38*	
	2.25		17.34					
48	27.74	25.4	21.10	2.06	2.04*	1.125	1.12*	
	28.64		22.37					
160	7.63	3.9	22.00	51.86	51.4†	0.06	0.05†	
	7.74		22.14					
34	3.97	0.0	19.00	2.00	2.00†	0.84	0.82†	
	4.05		18.85					
63	2.20	0.9‡	20.48	29.62	28.95†	0.32	0.30†	Sulphurets containing $\text{FeS}_2$ , $\text{PbS}$ , $\text{ZnS}$ (5 to 15 per cent.), $\text{Cu}_2\text{S}$ or $\text{Cu}$ , sulphides (2 to 11 per cent.).
	2.02		20.43					
71	7.16	3.4	18.07	63.06	62.4†	2.51	2.49†	
	7.20		18.50					
166	3.10	0.5‡	19.93	1.32	0.90†	0.21	0.20*	
	2.85		20.02					
57	.57	2.‡	.....	43.82	40.50†	0.92	0.90†	
	.48							
106	4.72	0.6	19.75	354.16	353.4†	0.48	0.42†	
	4.46		19.63					
107	10.93	7.3	22.10	84.38	83.7†	0.08	0.05†	
	10.90		21.62					
103	11.02	7.2	23.73	10.72	10.5†	0.06	0.05†	
	10.73		23.06					
105	.....	5.28	18.50	53.12	52.70†	0.02	trace.†	
			19.22					
99	.20	1.9‡	.....	4.20	4.00†	0.03	trace.†	
	.21							
75	.....	1.8	19.10	127.90	125.10†	0.31	0.30†	
			18.48					
40	10.3	7.	18.82	4.4	3.83*	3.01	3.00*	
			18.44					

\* By roasting method. † By niter-nails method. ‡ By scorification.

‡ Argol in place of niter where ore does not reduce enough lead.

*Corrected Assays.*

Corrected assays were made to compare the loss in the soda-litharge slag with the losses in the scorification and the crucible slags. Assays were therefore made on ores Nos. 63, 106 and 160, by the following methods:

I. Soda-litharge method.

II. Old crucible method (acid-slag).

## III. Scorification method.

The results obtained are given in Table X.

TABLE X.—*A Comparison of the Losses in the Slags Obtained in the Soda-Litharge Method, the Old Crucible Method and the Scorification Method. [Results given in ounces per ton.]*

Number of Ore.	The Soda-Litharge Method.					The Old Crucible Method.					The Scorification Method.				
	Silver Uncorrected.	Silver Correction for Slag.	Silver Correction for Cupel.	Total Silver.	Total Gold.*	Silver Uncorrected.	Silver Correction for Slag.	Silver Correction for Cupel.	Total Silver.	Total Gold.*	Silver Uncorrected.	Silver Correction for Slag.	Silver Correction for Cupel.	Total Silver.	Total Gold.*
68	28.70	.20	1.12	30.02	.32	27.80	.38	1.07	29.25	.32	27.25	1.50	1.75	30.50	.30
106	354.40	3.60	9.84	367.84	.42	†	†	†	†	†	353.55	3.35	9.75	366.25	.40
160	51.88	.82	1.84	54.54	.05	51.84	.50	1.42	53.76	.05	51.75	.70	1.81	54.26	trace.

\* No correction needed.

† No results were obtained, as the ore could not be assayed with an acid slag on account of the copper present.

Table X. shows that the new charge stands the comparison well and gives the uncorrected results nearest those of the corrected assay.

## SUMMARY.

The conclusions from the tests described above are: That it is necessary to determine the oxidizing power of niter with that substance and charge with which it is subsequently used; that the niter-method as above modified gives accurate results and is neither as long nor as tedious as the roasting method, nor is it as troublesome as the scorification method; it also gives a charge which does not boil over, and yields a lead button in the final assay which agrees closely in weight with the calculated quantity. Furthermore, a slag is produced which seems to combine the advantages of scorification with those of a crucible assay.

Experimental work is being continued, with the hope of obtaining an equally satisfactory method for the fire-assay of high-grade copper-matte, zinc-blende and very impure sulphide ores.

The work described in the foregoing paper was performed in the Havemeyer laboratories, Columbia University, New York City, and constitutes No. 82 in the series of "contributions" from these laboratories.

## Tests of Steel for Electric Conductivity, With Special Reference to Conductor-Rails.

BY J. A. CAPP, SCHENECTADY, N. Y.

(New York Meeting, October, 1908.)

FOR certain classes of electric railways a steel conductor is preferable to the older and more commonly used overhead trolley-wire. The third-rail presents a rather better appearance, because of the absence of an overhead structure; it is easily installed, cheaply maintained, presents a large area for conducting and collecting the current, and is therefore particularly suitable for high speed and heavy service. With costs calculated on the basis of equal conductivity in rail and trolley-wire, the third-rail construction is cheaper than the overhead trolley. But the average interurban road will use a trolley-wire of considerably less conductivity than would be obtained with the smallest size steel rail (about 60 lb. per yard) that would ordinarily be used, and here the first cost of trolley construction would generally be less than that of the third-rail. While no third-rail installation has yet been in operation long enough to give figures of value, it would appear that the cost of maintenance of the third-rail construction should be less than that of the overhead trolley. This consideration, together with that of sightliness and adaptability (particularly in the case of terminals, yards and very heavy or high-speed service), will frequently offset the higher cost of the third-rail construction and make it the preferable means of conducting the current from the generator to the car-motor.

For the first third-rail installations, old track rails were used when obtainable, and these old rails were supplemented with new rails of standard T-section and composition. With the coming of the very heavy, high-speed service during the past few years, the resistance of these old and new standard T-rails was found to be so high that they would not carry the high currents necessary without too great a drop in the line-potential. The rails would, therefore, have to be supplemented with addi-

tional copper feeders, but as this would be an expensive way to overcome the trouble, rails of higher specific electric conductivity were sought.

Because of the lack of specific data on the relation between conductivity and composition of mild steels, specifications for conductor-rails have usually been based on the fact that the conductivity of a metal is generally more or less directly in proportion to its purity. In most cases the purity of the iron specified for such rails has been so high that not only was it difficult to obtain, but the iron was also correspondingly of high price. One of the factors governing the choice between a third-rail and a trolley-wire is the relative price of steel and copper, allowance being made for the difference in conductivity. Hence, a balance must be struck between high conductivity (which is equivalent to saying a high degree of purity or freedom from the usual metalloids associated with iron) and the cost of producing the steel of the composition necessary for the conductivity required.

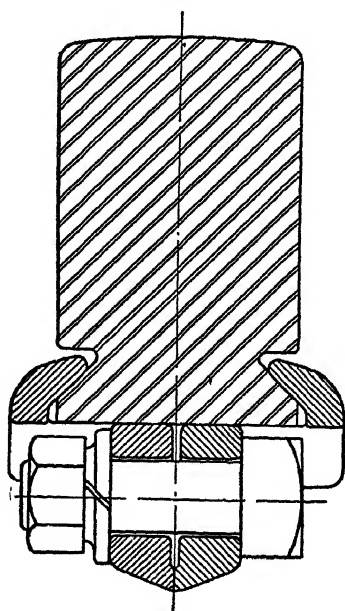
With the object of drafting a rational specification for such third-rail steels, a series of tests was started last spring, in the testing laboratory of the General Electric Co., on steels of as wide a range of composition as could be obtained from the steel makers. The results of these tests form the basis of this paper.

On entering the market with an order for rails of special composition, difficulty was encountered in interesting rail-makers, and the reason became apparent. The mills which make the standard T-rails, or standard rails of other sections (such as the girder and grooved rails), have generally been designed with the object of making such rails only, and practice throughout has been standardized to the greatest possible extent, both in the manipulation of the steel and in its manufacture in the furnace or converter. Hence, an order, even of considerable extent, for rails of special composition, has so upset the routine work that it was considered a nuisance to be endured, rather than a desirable business to be sought for. Obviously, then, recourse must be had to mills where such special compositions can be handled economically.

The T-section of rail was the outcome of designs intended to present the greatest life and strength of the section, with the

least weight of the metal, conditions necessarily to be met in a running-rail. But the conditions for the third- or conductor-rail are different; here, there must be provided sufficient surface for the collection of the current, and cross-section ample to carry the current without an undue drop in the line-potential. The strength of the section is of little moment, and any section which is easily installed in an insulator is satisfactory. This permits the use of sections rectangular, or nearly so, which may be rolled easily in any mill equipped for the rolling of merchant-bar shapes of reasonably heavy weight, without any

FIG. 1.



CROSS-SECTION OF A NEW CONDUCTOR RAIL, DESIGNED BY MR. W. B. POTTER.

change in equipment or practice beyond the provision of grooved rolls of the necessary shape. Mills rolling merchant-bar or structural steel, to-day, are generally equipped to make steel in the open-hearth furnace, which readily lends itself to the making of steel of the special composition demanded by the electrical engineers for the third- or conductor-rail.

Having these facts in mind, a section of a conductor-rail has been designed by Mr. W. B. Potter, Chief Engineer of the Railway Department of the General Electric Co., which, when

2.5 in. wide by 4 in. high, will weigh about 98 lb. to the yard. This shape, which is shown in Fig. 1, may be easily rolled in any merchant-bar mill heavy enough to attempt sections of this weight. A dovetail at the bottom provides an easy

TABLE I.—*Resistance and Composition of Steel.*

Serial Number.	Specific Resistance.		Conductivity.	Resistance.	Percentage Composition.							
	Microhms Per Cm Cm <sup>2</sup> .	Temp., ° C.			Cu. = 1.	C.	Mn.	P.	S.	Si.	Total, Not Fe.	P. + S. + Si.
1	22.72	19°	7.58	13.20	0.33	1.27	0.09	0.05	0.05	1.79	0.19	
2	20.90	20°	8.27	12.12	0.17	1.09	0.09	0.05	0.004	1.404	0.144	
3	21.29	25°	8.27	12.09	1.40	0.222	0.01	0.020	0.082	1.734	0.112	
4	19.87	19°	8.65	11.55	0.20	0.95	0.10	0.08	0.05	1.38	0.230	
5	19.80	19°	8.68	11.51	0.43	0.77	0.10	0.04	0.066	1.406	0.206	
6	19.80	19°	8.69	11.51	0.36	0.80	0.10	0.04	0.047	1.347	0.187	
7	19.81	20°	8.69	11.51	0.22	1.08	0.10	0.05	0.06	1.510	0.210	
8	19.69	19°	8.73	11.44	0.74	0.58	0.043	0.036	0.20	1.599	0.279	
9	18.95	25°	9.29	10.76	1.61	0.147	0.015	0.018	0.092	1.882	0.125	
10	18.17	19°	9.46	10.56	0.41	0.72	0.039	0.041	0.11	1.32	0.190	
11	17.27	19°	9.96	10.04	0.36	0.87	0.03	0.09	0.04	1.44	0.210	
12	17.10	19°	10.06	9.94	0.37	0.73	0.09	0.04	0.06	1.29	0.190	
13	17.10	19.5°	10.06	9.94	0.23	0.80	0.016	0.033	0.016	1.095	0.065	
14	16.96	19°	10.14	9.86	0.30	0.95	0.063	0.01	0.01	1.333	0.083	
15	16.95	19.5°	10.14	9.86	0.29	0.99	0.084	0.01	0.01	1.384	0.104	
16	16.32	19°	10.55	9.48	0.23	0.89	0.058	0.01	0.005	1.193	0.073	
17	16.25	19.5°	10.59	9.44	0.26	0.83	0.053	0.01	0.004	1.157	0.087	
18	16.21	20°	10.62	9.42	0.28	0.65	0.083	0.06	0.05	1.123	0.193	
19	16.09	19°	10.69	9.36	0.22	0.68	0.077	0.07	0.05	1.097	0.197	
20	16.09	19°	10.69	9.36	0.16	0.66	0.074	0.030	0.014	0.938	0.118	
21	15.32	19°	11.24	8.90	0.33	0.49	0.068	0.05	0.02	0.958	0.138	
22	14.57	19.5°	11.82	8.46	0.31	0.45	0.10	0.04	0.026	0.926	0.166	
23	14.49	20°	11.88	8.42	0.25	0.41	0.10	0.04	0.03	0.83	0.170	
24	14.73	23.5°	11.88	8.42	0.144	0.46	0.09	0.08	tr.	0.774	0.170	
25	14.62	23.5°	11.96	8.36	0.188	0.48	0.09	0.08	tr.	0.83	0.170	
26	14.15	19°	12.17	8.22	0.22	0.56	0.024	0.34	tr.	0.838	0.058	
27	14.03	19°	12.26	8.16	0.192	0.57	0.024	0.34	tr.	0.82	0.058	
28	13.86	19°	12.41	8.06	0.16	0.48	0.091	0.04	0.01	0.781	0.144	
29	13.83	19.5°	12.44	8.04	0.10	0.55	0.08	0.05	0.024	0.804	0.154	
30	13.80	19°	12.57	8.02	0.14	0.41	0.11	0.05	0.009	0.719	0.169	
31	13.67	19°	12.58	7.95	0.23	0.48	0.024	0.01	0.023	0.767	0.057	
32	13.64	19°	12.61	7.93	0.24	0.57	0.029	0.01	0.003	0.850	0.042	
33	13.90	24°	12.63	7.92	0.10	0.25	0.04	0.02	0.05	0.46	0.110	
34	13.31	19°	12.92	7.74	0.25	0.37	0.04	0.03	0.018	0.708	0.088	
35	13.30	19.5°	12.94	7.73	0.23	0.49	0.024	tr.	0.004	0.748	0.028	
36	13.27	19°	12.97	7.71	0.19	0.37	0.09	0.05	0.01	0.71	0.150	
37	13.25	19°	12.99	7.70	0.27	0.41	0.024	0.01	0.001	0.715	0.035	
38	13.18	19°	13.05	7.66	0.28	0.28	0.027	0.034	0.04	0.661	0.111	
39	13.18	19°	13.05	7.66	0.07	0.40	0.03	0.07	0.013	0.633	0.163	
40	13.07	19°	13.16	7.60	0.28	0.42	0.022	0.04	0.008	0.770	0.070	
41	12.87	20°	13.27	7.48	0.16	0.38	0.08	0.04	0.009	0.669	0.129	
42	12.73	20°	13.52	7.40	0.15	0.45	0.011	0.033	tr.	0.644	0.044	
43	12.69	19°	13.55	7.38	0.19	0.21	0.025	0.04	0.034	0.499	0.099	
44	12.53	19°	13.74	7.28	0.215	0.22	0.051	0.113	.....	0.599	0.164	
45	11.01	19°	15.63	6.40	0.05	0.19	0.054	0.059	0.03	0.383	0.143	



means of securing the rails by fish plates of special forms, and any of the common forms of bond may easily be applied. Ordered in lots of a thousand tons or more, such a rail should cost no more than a plain rectangle of equal weight.

To provide steels of sufficient range to give some indication of the relation between conductivity and composition, we obtained from as many of the steel makers as we could interest, samples representing all of their common products. Some of the makers kindly provided us also with samples of special steels made in crucible charges. The samples were mainly forged or rolled bars of from 1 to 2.5 in. in diameter, and from them were turned bars of 0.75 in. in diameter, or 1 in. in diameter by 24 in. in length, on which resistivity was determined. The chips from the finishing cuts were collected and these furnished the samples for analysis. Also, we cut similar bars from the heads of T-rails which were to be found in the yard tracks.

Table I. states the electrical resistance and the chemical

TABLE II.—*Resistance and Composition of Iron.*

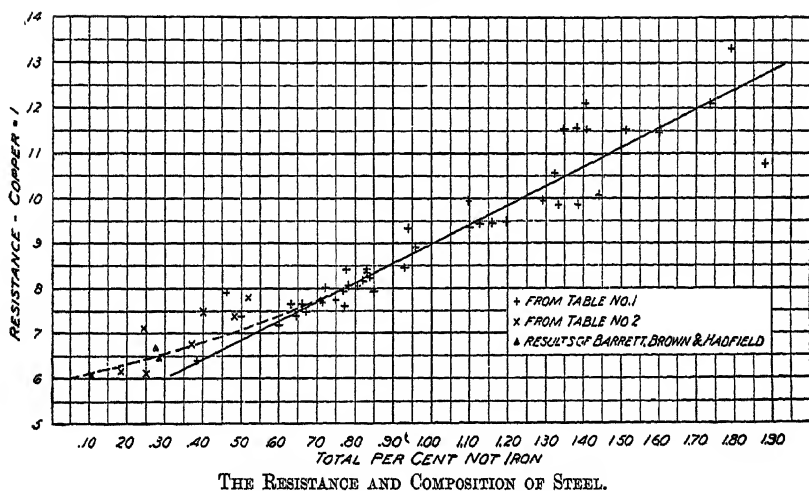
Serial Number.	Specific Resistance.		Conductivity.	Resistance.	Percentage Composition.							
	Microhms Per $\frac{\text{Cm.}}{\text{Cm}^2}$	Temp., ° C.	Matthies- sen Standard.	Cu. = 1.	C.	Mn.	P.	S.	Si.	Total, Not Fe.	P. + S. + Si.	
46	13.80	25.5	12.78	7.82	0.150	0.068	0.130	0.020	0.150	0.518	0.300	
47	13.82	26.	13.37	7.48	0.150	0.064	0.036	0.020	0.130	0.400	0.186	
48	13.10	26.	13.50	7.41	0.160	0.074	0.120	0.027	0.100	0.481	0.247	
49	12.54	25.5	14.07	7.11	0.080	nil.	0.130	0.008	0.024	0.242	0.162	
50	11.92	25.5	14.80	6.76	0.170	0.027	0.074	0.022	0.077	0.370	0.173	
51	10.82	24.	16.21	6.17	0.058	0.10	0.014	tr.	0.012	0.184	0.026	
52	10.80	25.5	16.34	6.12	0.160	0.018	0.049	0.011	0.015	0.252	0.075	
LSS	11.40	17.	15.00	6.68	0.050	0.180	0.013	0.011	0.020	0.274	0.044	
B	11.00	17.	15.57	6.44	0.030	0.036	0.065	0.016	0.140	0.287	0.221	
SCI	10.35	17.	16.55	6.06	0.028	tr.	0.004	0.005	0.070	0.107	0.079	

composition of 45 samples of steel, and Table II. similar data on 7 samples of wrought, or refined, iron, together with results on 3 samples of iron reported by Barrett, Brown and Hadfield in a paper referred to later in this article. The samples were numbered serially in the order of their resistance, No. 1 being of the highest resistance. Samples Nos. 1, 2, 4, 7, 11 and 12 are standard T-rails from several well-known makers,

while Nos. 24 and 25 are cut from the 100-lb. T-rail used for the conductor on the Aurora, Elgin and Chicago Railroad. No. 45 is a T-rail used by the Underground Electric Railways Co. of London, and ordered from a steel maker in Westphalia. Samples Nos. 46, 47 and 48 are ordinary refined bar-iron, while Nos. 49 and 50 are special brands of refined bar-iron sold for use in staybolts and similar work. Nos. 51 and 52 are Swedish and Norway irons, respectively.

The resistance is expressed in the three ways in which such results are frequently stated. The first column gives the specific resistance or resistivity of a section one square centimeter

FIG. 2.



in cross-section and one centimeter long, expressed in microhms (ohms  $\times 10^{-6}$ ) and at the temperature given in the second column. In the third column is given the conductivity (on the basis of Matthiessen's determination of the resistivity of pure copper), *i.e.*, the ratio of the resistivity of copper to that of the sample, expressed as a percentage. The fourth column gives the resistance of the samples compared with Matthiessen's copper as unity (the figures in the fourth column are the reciprocals of those in the third column multiplied by 100). The second half of the table gives the percentages of the usual elements to be found in steel or iron, the total percentage of these elements, and the sum of the percentages of phosphorus, sulphur and silicon.

The coefficient of change of resistance with change of temperatures is practically the same for copper and for ordinary steel and iron, hence, even though resistivities are determined at different temperatures, the conductivities or comparative resistances derived therefrom are directly comparable within limits close enough for the purpose of this discussion, without actual determinations of the temperature-coefficient in each case, and the reduction of results to a basis of common temperatures. Therefore, in all succeeding reference to results, the resistance will be expressed as in the fourth column, *i. e.*, the ratio of resistivity of sample to that of Matthiessen's copper.

The resistance of steel, as of other alloys, might be expected to vary more or less directly with its purity. Therefore, in the curve shown in Fig. 2 are plotted resistances and total percentages of metalloids. The curve was first drawn from the data in Table I. and was approximately a straight line, but on adding the results obtained for refined iron from Table II., the lower part of the curve was found to bend up as shown by the broken line. This would indicate that the increase in resistance, due to increasing percentages of metalloids, becomes

TABLE III.—*Resistance of Steel. Variation with Manganese.*  
(Carbon from 0.17 to 0.23 Per Cent.)

Sample Number.	Manganese.	Resistance.	Carbon.	P. + S. + Si.
	Per Cent.		Per Cent.	Per Cent.
2	1.09	12.12	0.17	0.144
4	0.95	11.55	0.20	0.230
7	1.08	11.51	0.22	0.210
13	0.80	9.94	0.23	0.065
16	0.89	9.48	0.23	0.073
19	0.68	9.36	0.22	0.197
25	0.48	8.36	0.188	0.170
26	0.56	8.22	0.22	0.058
27	0.57	8.16	0.192	0.058
31	0.48	7.95	0.23	0.057
35	0.49	7.73	0.23	0.028
36	0.37	7.71	0.19	0.150
43	0.21	7.38	0.19	0.099
44	0.22	7.28	0.215	0.164

gradually greater up to about 0.75 per cent. of metalloids, after which it follows a straight line up to about 2 per cent. of metalloids; beyond this latter point our samples did not extend.

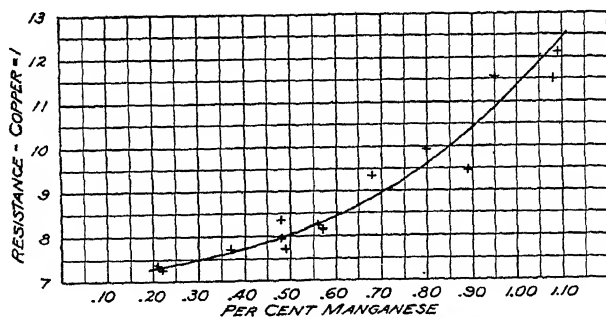
By the selection of samples of a composition varying in one element only, an attempt was made to define separately the influence of the several elements upon the resistance of steels. In Table III. are listed the samples having carbon between

TABLE IV.—*Resistance of Steel. Variation in Manganese.*  
(Carbon from 0.27 to 0.33 Per Cent.)

Sample Number.	Manganese.	Resistance.	Carbon.	P. + S. + Si.
	Per Cent.		Per Cent.	Per Cent.
1	1.27	13.20	0.33	0.190
14	0.95	9.86	0.30	0.083
15	0.99	9.86	0.29	0.104
18	0.65	9.42	0.28	0.193
21	0.49	8.90	0.33	0.138
22	0.45	8.46	0.31	0.166
37	0.41	7.70	0.27	0.035
38	0.28	7.66	0.28	0.111
40	0.42	7.60	0.28	0.070

0.17 and 0.23 per cent., manganese being the principal variable; and in Table IV. are listed samples with manganese variable and carbon constant at from 0.27 to 0.33 per cent. These results are plotted on the curves shown in Figs. 3 and 4,

FIG. 3.



RESISTANCE OF STEEL. VARIATION IN MANGANESE.  
(Carbon from 0.17 to 0.23 per cent.)

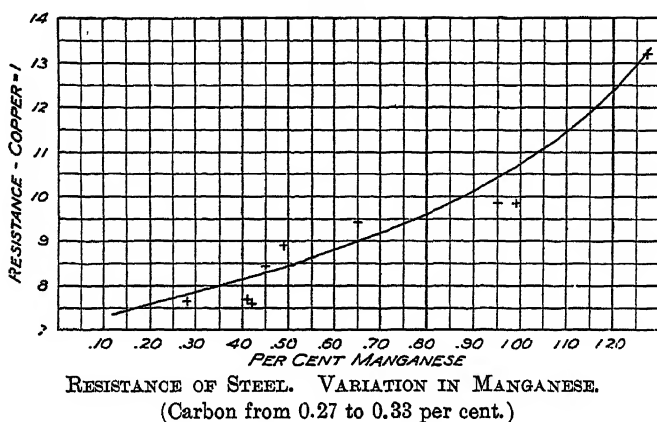
which show that the effect of manganese in increasing resistance gradually increases with the percentage of manganese present, within the limits represented by these samples.

Messrs. Barrett, Brown and Hadfield,\* in the study of the

\* "On the Electrical Conductivity and Magnetic Permeability of Various Alloys of Iron," by W. F. Barrett, W. Brown and R. A. Hadfield, in *Transactions of the Royal Dublin Society*, vol. vii., series 2, Part IV.

influence of manganese on resistance, used steels containing from 0.5 to 18.5 per cent. of manganese, and found the resistance to increase at first very rapidly, with constantly increasing percentage of manganese, then more and more slowly, until 7 per

FIG. 4.



cent. of manganese, after which a further increase in the percentage of manganese produces little or no increase in resistance. Le Chatelier\* also gives figures showing the influence of manganese on resistance, but he has reported five samples only,

TABLE V.—Resistance of Steel. Variation in Carbon.  
(Manganese from 0.15 to 0.28 Per Cent.)

Sample Number.	Carbon.	Resistance.	Manganese.	P. + S. + Si.
	Per Cent.		Per Cent.	Per Cent.
3	1.40	12.09	0.222	0.112
9	1.61	10.76	0.147	0.125
38	0.10	7.92	0.25	0.110
38	0.28	7.66	0.28	0.111
43	0.19	7.38	0.21	0.099
44	0.215	7.28	0.22	0.164
45	0.05	6.40	0.19	0.143

and these are of such high carbon-content as to be of no service to us in the study of steels available for conductor-rails.

To determine the influence of carbon, we have selected, in Table V., the steels with manganese constant at from 0.15 to 0.30

\* "Sur le Resistance Electrique des Alliages," par Mon. H. Le Chatelier, in "Contribution à l'Etude des Alliages," published by the committee on alloys of the Société d'Encouragement pour l'Industrie Nationale.

per cent., and with carbon as the principal variable. In Table VII. we have stated results given by Le Chatelier in the paper previously mentioned, and in Table VIII. the results of Barrett, Brown and Hadfield taken from the paper previously

TABLE VI.—*Resistance of Steel. Variation in Carbon.*  
(*Manganese from 0.4 to 0.49 Per Cent.*)

Sample Number.	Carbon.	Resistance.	Manganese.	P. + S. + Si.
	Per Cent.		Per Cent.	Per Cent.
21	0.33	8.90	0.49	0.138
22	0.31	8.46	0.45	0.166
23	0.25	8.42	0.41	0.170
24	0.144	8.42	0.46	0.170
25	0.188	8.36	0.48	0.170
28	0.16	8.06	0.48	0.144
30	0.14	8.02	0.41	0.169
31	0.23	7.95	0.48	0.057
35	0.23	7.73	0.49	0.028
37	0.27	7.70	0.41	0.035
39	0.07	7.66	0.40	0.163
40	0.28	7.60	0.42	0.070
42	0.15	7.40	0.45	0.044

quoted. All of these figures are plotted on the curve shown in Fig. 5, where the solid straight line represents closely the results of Le Chatelier, while the broken line follows more nearly the average of the combined figures from all three

TABLE VII.—*Resistance of Steel. Influence of Carbon.*  
(*Results of M. Le Chatelier.*)

Resistance.		Composition.		
Microhms.	Cu. = 1.	C.	Mn.	Si.
		Per Cent.	Per Cent.	Per Cent.
10	5.78	0.06	0.13	0.05
12.5	7.22	0.20	0.15	0.08
14	8.10	0.49	0.24	0.05
16	9.25	0.84	0.24	0.13
18	10.40	1.21	0.21	0.11
18.4	10.64	1.40	0.14	0.09
19	11.00	1.61	0.13	0.08

sources. With uniformly increasing carbon, the resistance at first rises very rapidly, but the rate of increase gradually drops until it reaches a straight line at about 0.2 per cent. C, which continues up to limits of the carbon listed. In all cases the

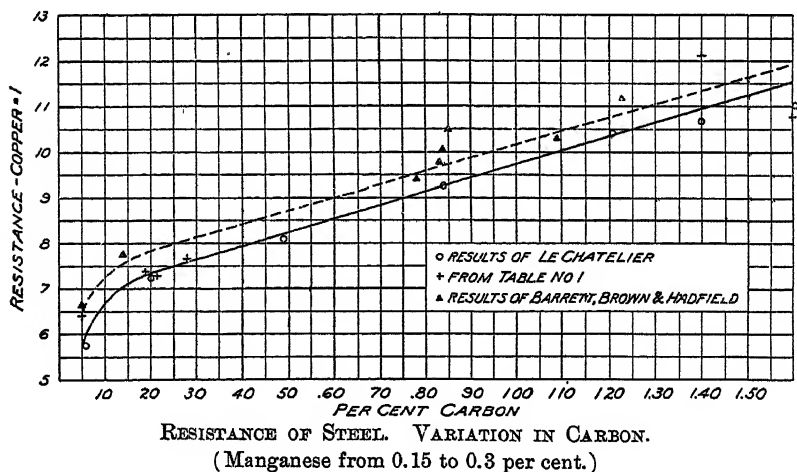
steel is supposed to be unhardened, as it has been shown by Barus and by Le Chatelier that hardened steels may have double or even treble the resistance found in the annealed state. Table VI. selects samples, with manganese constant at

TABLE VIII.—*Resistance of Steel. Variation in Carbon. Results of Barrett, Brown and Hadfield. Temperature 17° C.*

Sample Mark.	Resistance.		Composition.		
	Microhms.	Cu. = 1.	Carbon.	Manganese.	Silicon.
			Per Cent.	Per Cent.	Per Cent.
1392G	19.1	11.19	1.23	0.14	0.12
1392L	17.6	10.31	1.09	0.32	0.17
1392A	17.9	10.49	0.85	0.32	0.17
1392B	17.2	10.07	0.84	0.18	0.20
1392I	16.7	9.78	0.83	0.25	0.06
1392H	16.1	9.43	0.78	0.10	0.10
1166A	18.4	7.85	0.14	.....	0.08

from 0.4 to 0.5 per cent. and carbon variable; these results are plotted on the curve shown in Fig. 6, but the range of the carbon percentages is too small to permit the drawing of any

FIG. 5.



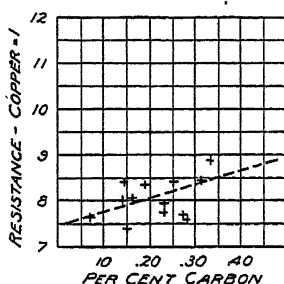
curve. The broken line is drawn parallel to the straight portion of the line in the curve shown in Fig. 5.

The elements phosphorus, sulphur and silicon were not present in sufficient quantity in any of the samples tested to permit

us to draw any curves showing their influence upon the resistance. In both the papers previously quoted, the influence of silicon, when present in large quantity, is studied, but the number of samples tested is too small to warrant any conclusions that are really definite. In commercial steels the percentages of all three of these elements is so small that their effect on resistance may generally be neglected.

A study of the tables and curves given in this paper shows that manganese preponderates in influencing the resistance of steels, and that for lowest resistivity, this element must be present in very small quantity, much smaller than is usual in merchant- or structural-steels. While all the other elements must be present only in very small percentages, so great is the pre-

FIG. 6.



RESISTANCE OF STEEL. VARIATION IN CARBON.  
(Manganese from 0.4 to 0.5 per cent.)

ponderance of the influence of manganese, that they may be tolerated in quantities which the steel makers would consider reasonable, without unduly increasing the resistance.

For a satisfactory third-rail, the lowest possible resistance (from 6 to 6.5 times that of copper?) is not necessary; and the great cost of making such extremely pure steel is not warranted. In fact, such extremely pure steels would probably be so soft that the frictional wear of the collecting shoe would be excessive and the life of the rail in service unduly short. Assuming, then, that a rail made from steel having a resistance not greater than eight times that of copper (13.8 microhms at 20° C) would be desirable for conductor-rails, the figures tabulated would seem to indicate that the following extreme composition would be permissible:—



	Per Cent.
Carbon up to, . . . . .	0.20
Manganese up to, . . . . .	0.40
Phosphorus up to, . . . . .	0.06
Sulphur up to, . . . . .	0.06
Silicon up to, . . . . .	0.05

This composition, however, would be extreme, and any overstepping of bounds might result in too great resistance; therefore, for resistance up to eight times that of copper, the specified analysis should be:—

	Per Cent.
Carbon not to exceed, . . . . .	0.15
Manganese not to exceed, . . . . .	0.30
Phosphorus not to exceed, . . . . .	0.06
Sulphur not to exceed, . . . . .	0.06
Silicon not to exceed, . . . . .	0.05

This latter composition is one which could be made easily in any open-hearth furnace and it should present no difficulty in rolling to a shape suitable for conductor-rails, such as that shown in Fig. 1. In fact, steel of this composition has been successfully rolled into sheets as thin as 0.014 in., and was for a long time a standard product of a large sheet-mill.

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### Biographical Notice of William Earl Dodge.

BY JAMES DOUGLAS, NEW YORK CITY.

(New York Meeting, October, 1903.)

ONCE—and fortunately only once—before this occasion, was it my sad duty to express the loss we had sustained in the death of an eminent member. Dr. T. Sterry Hunt\* was a student of chemistry who, by his investigations into the properties of the elementary bodies, had assisted eminently in the progress of the technical arts, and, more than any American of his generation, had helped to lay the foundations of chemical geology. William Earl Dodge, whose death we deplore to-day, occupied a rank not less important in the army of industrial workers

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\* See "Biographical Notice of Thomas Sterry Hunt," *Trans.*, xxi., 400.

than that of the scientific investigator. For unless there were men of wealth, willing to take the risks of applying the laws which the student unravels, and of putting to the test of practice his theoretical hypotheses, industrial progress would be slow, and certainly mining and metallurgy would languish. Moreover, the student, or even the practical mill-man, furnace-man, or miner, is rarely a good business man. In his enthusiasm he overlooks the prime condition of success—ultimate profit. He neglects that minute analysis of expenditure and receipts, and that classification and standardization of operations which alone can enable anyone to know whether he is advancing towards, or receding from, the goal which every practical worker must keep in view, namely, pecuniary gain. This motive may seem ignoble; nevertheless, profit is, in the last resort, the inevitable object of pursuit in our profession; and, after all, profit, legitimately earned, is the measure of our usefulness to our fellows, and their recognition of the service.

Another habit, or acquired faculty, which the business man possesses to a higher degree than the student, is that of judging character, and thus selecting wisely the men who are to aid, in its different departments, towards bringing an undertaking to successful fruition.

I am, therefore, not far astray in assigning to the man of business, under our modern industrial system, a status little inferior to that of the man of science and purely technical training. From this standpoint, Mr. Dodge was certainly entitled to be enrolled as one of our members, for of that important class of men which, in this country, thus works in co-operation with us, he was one of the most eminent.

He was a man of marvelous versatility, and while he exerted tireless energy in the furtherance of the innumerable unselfish projects to which he devoted not only his money, but his time, he was nevertheless as punctual in answering the call, and as assiduous in fulfilling the duties of the many business enterprises in which he was interested, as though his whole soul had been concentrated on commercial pursuits. To those of us who were associated with him, and knew him best, the wonder was how he possibly could do so many things and do them all so well. That he should have invariably acted honorably was inevitable, because his intentions were never other than genèr-

ous and true; but that he should have reached, so rapidly, correct conclusions on the multitudinous subjects that demanded prompt decision, seemed to be the result of instinct, derived from long habits of healthy thought, rather than of conscious processes of reasoning.

It does not fall within my province to describe his public life; yet one cannot refrain from recalling his invaluable services to the Nation, as a member of the Sanitary Commission during the Civil War; the courageous part he played during the conscription-riots in this city; the devotion of time and money which he so liberally exhibited in the cause of International Arbitration, of the Committee on which he was chairman; the debt which the financial world owes to him for his fight for sound currency in the monetary conference at Indianapolis; the life-long work which he did for young men, first as a young member, then as President, and ultimately as Chairman of the New York Branch, of the Young Men's Christian Association; his efforts (not altogether unsuccessful) as President of the "Evangelical Alliance," to lift the Christian church above the narrow confines of denominationalism to loftier standards of freedom and to more united action; his contribution towards higher education in the aid he extended to the N. Y. Teachers' College; and to technical education, in the leading part he took in the foundation and support of the Columbia School of Mines, the American Museum of Natural History, the Metropolitan Museum of Art, and the New York Botanical Garden. New York City and the country have to thank him also for introducing to them that inimitable public teacher, Professor Albert S. Bickmore. It may be added, that he was a Trustee of the Carnegie Institution, for the encouragement of scientific research, the work of which has but just begun.

To all these, and many other public services, he devoted unstintingly both time and money; and yet he never neglected his business investments and interests,—so elastic and comprehensive can become a liberal purse and the energies of a noble life!

These were but a few of those of Mr. Dodge's labors of which the public had cognizance. In his private benefactions, he followed the Master's injunction, and let not his left hand

know what his right hand did. As one of the many who have experienced, in trying times, his generous aid and sound advice, I can say that no man ever practiced more skilfully than he, the difficult art of extending help without creating a sense of degradation in the recipient.

It was a natural consequence that, as a representative of the third generation of a firm of metal-merchants, Mr. Dodge should take an active interest in mining, and in metallurgical undertakings. But I think that the position which he and his eminent partner, Mr. D. Willis James, have assumed, in transferring their energies and capital from mercantile pursuits to cognate mining and metallurgical enterprises, when the drift of events in this country foreshadowed the decline of our importing and the growth of our producing-capacity, is worthy of notice as almost unique. Mr. Dodge had always taken his share in the risks of mining. He was a shareholder in the old Minnesota mine at Lake Superior. He was deeply interested in the North Star gold-mine of Grass Valley, Cal.; and, long before California was reachable by rail, he lost money in testing the value of the Temescal tin-mines; in fact, he and his partners were led by their interests in tin to investigate every discovery on this continent of that widely distributed, yet sparingly concentrated, metal. Moreover, he was a director of the Lackawanna Iron & Coal Co., and its successor, the Lackawanna Steel Co., and of the Ansonia Copper Co. and the Ansonia Clock Co. It has been only during the last twenty-two years, however, that the special direction of business ability and capital towards the development of the copper resources of the southwest has made the name of Phelps, Dodge & Co. a household word in that section of the country, as well as in northwestern Mexico. Instead of scattering their investments, their distinct policy, which Mr. Dodge consistently advocated, has been to concentrate their efforts on the development of the mines of a given area, and to use every means which money could command, and technical skill and prudent business management could suggest, for economically and yet liberally opening their copper-mines, prosecuting their exploitation, and treating the ore produced. These operations have been conducted as private business enterprises. No taint of stockjobbing has ever been attached to them, and their securities are

not on the market. Even upon their railroad system of over four hundred miles no bonds have been issued. Their purpose and effort have been to conduct their mines, and the railroads which were built as subsidiary to the mines, on the same principles which they had applied to their mercantile affairs. I think that, especially in these days, this example is one of no slight importance; and I know that whatever success has attended it, has been due in great measure to Mr. Dodge's consistency and tremendous courage; for I need not disguise the fact that the carrying through of such a policy by a single firm involved, at times, no little risk.

Nor have these enterprises been conducted in any selfish spirit. The railroads of this firm,—the El Paso & Southwestern, the Morenci Southern, and the Nacozari railroad,—were laid out primarily to link together the several mines of the firm with central metallurgical works, now being erected at the junction of their United States and Mexican railroads. But these railroads were also built to secure, not only to their own mines and metallurgical works, but to those of their neighbors, access at low rates of freight to El Paso, the most important competitive railroad-point in the southwest. The recent rapid development in mining interests of that section of the country is due, in no small degree, to the liberal policy of these men.

An evidence of Mr. Dodge's open-mindedness in business matters is afforded by his efforts, in connection with Mr. Keyser, of Baltimore, to create the Copper-Producers' Association, under Mr. John Stanton as statistician. From its inception, it was an international association of the large copper-producers, organized to collect and issue information as to the output, imports and exports of copper, with the view of giving to the trade data on which to secure a steady market. Neither Mr. Dodge nor his partners ever had the slightest objection to publishing the statistics of any of their mining operations. But in course of time some members of this Association preferred that their individual output be not known, and insisted on having the reported production classed under groups. Ultimately, the objection to publicity became so strong that the Association was dissolved. Mr. Dodge, although educated under the older maxims of trade, had become convinced that more was lost than gained by secrecy; that, even in technical matters, no one

is so wise and so well informed that he can afford to dispense with all that his neighbors may know; and that the law of interchange of ideas is as inviolate as the law of interchange of new products, if healthy progress is to be maintained.

Concerning Mr. Dodge's relations to the American Institute of Mining Engineers, I cannot do better than to let its Secretary speak, as he does in the following letter:

NEW YORK, October 12, 1903.

MY DEAR DR. DOUGLAS:

In the Biographical Notice of the late William E. Dodge, which you are to present to-morrow, at the first session of the New York meeting, I beg you to include my individual and official testimony concerning his relations with the Institute.

Long before he became a member, we had had ample opportunity of knowing how heartily he appreciated the purpose and the work of the Institute. In 1899, he gave new evidence of this feeling by extending to the Council exceptionally liberal terms as to rent, etc., for the occupancy of rooms in the new fire-proof building of Phelps, Dodge & Co., then just completed. At that time, the office of the Secretary, rented, at exceedingly low rates, from Cooper, Hewitt & Co. (through the generosity of Abram S. Hewitt and Edward Cooper), had become inadequate for its increased business and the accumulating library; and it had become imperatively necessary to find, elsewhere, accommodations, both larger and safer. The Council, therefore, accepted the very liberal offer of Phelps, Dodge & Co., realizing that its moderate terms were largely dictated by friendship to the Institute.

At the same period, as if to show that he desired a relation more sympathetic than that of landlord, Mr. Dodge caused himself to be proposed (and was in 1900 duly elected) as member of the Institute, and from that time to this, his generous desire to promote its welfare has been made known in innumerable ways. Privileges and concessions which, as landlord, the firm was not bound to grant, have been freely extended to us, in the spirit which Mr. Dodge entertained, and his partners shared. If we had been our own landlords, we could not have treated ourselves better than they have treated us.

This official testimony I beg to reinforce with my personal

witness, based upon an acquaintance and friendship of many years, to the unvarying and unfailing generosity and sympathy of Mr. Dodge.

Yours truly,

R. W. RAYMOND,

*Secretary.*

I think I have demonstrated that there are no more useful or worthy members of our Association than business men of Mr. Dodge's type. Of one thing I am certain: that nothing I have said, or that anyone can say, will adequately express his sagacity in business affairs, and yet, withal, the breadth of his human sympathies, his exquisite courtesy and the depth of his Christian convictions. To reconcile the maxims of our Master with the trying requirements of mercantile life, is as difficult in the twentieth century as it was in the first. If anyone has succeeded in doing so, it was William Earl Dodge.

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### Relative Elimination of Impurities in Bessemerizing Copper-Matte.

BY W. RANDOLPH VAN LIEW, GLOBE, ARIZONA.

(New York Meeting, October, 1903.)

IN determining the relative rate and in finding the point where different impurities contained in copper-matte are eliminated during the process of a converter-blow, the following results were obtained:

A converter was selected which was starting on its second charge. The first charge after lining had finished its copper "hot," and consequently no copper was adhering to the sides of the lining. All the copper and granulated slag from the previous charge were dumped, thus removing any possibility of "salting" the matte to be tested.

The converter worked fast and well during the entire test. Periods of ten minutes were selected. The converter was brought from the stack to secure each sample of matte to be

analyzed, and only the time during which air was being forced through the charge was counted.

At the end of forty minutes' actual blowing, the matte was up to "white metal," the point at which the last skimming takes place; that is, matte of approximately 76.4 per cent. copper. From here, of course, average samples of the contents of a converter are impossible, since (from this skimming-point up to finished copper) the contents of a converter consist of constantly varying proportions of matte and copper, which, when a converter is brought from the stack and the blast-pressure turned off, settle according to their specific gravity. When the charge is completed a granulated sample of finished copper is obtained and assayed. Accurate chemical analyses of these equal-period samples of matte and copper gave the following results:

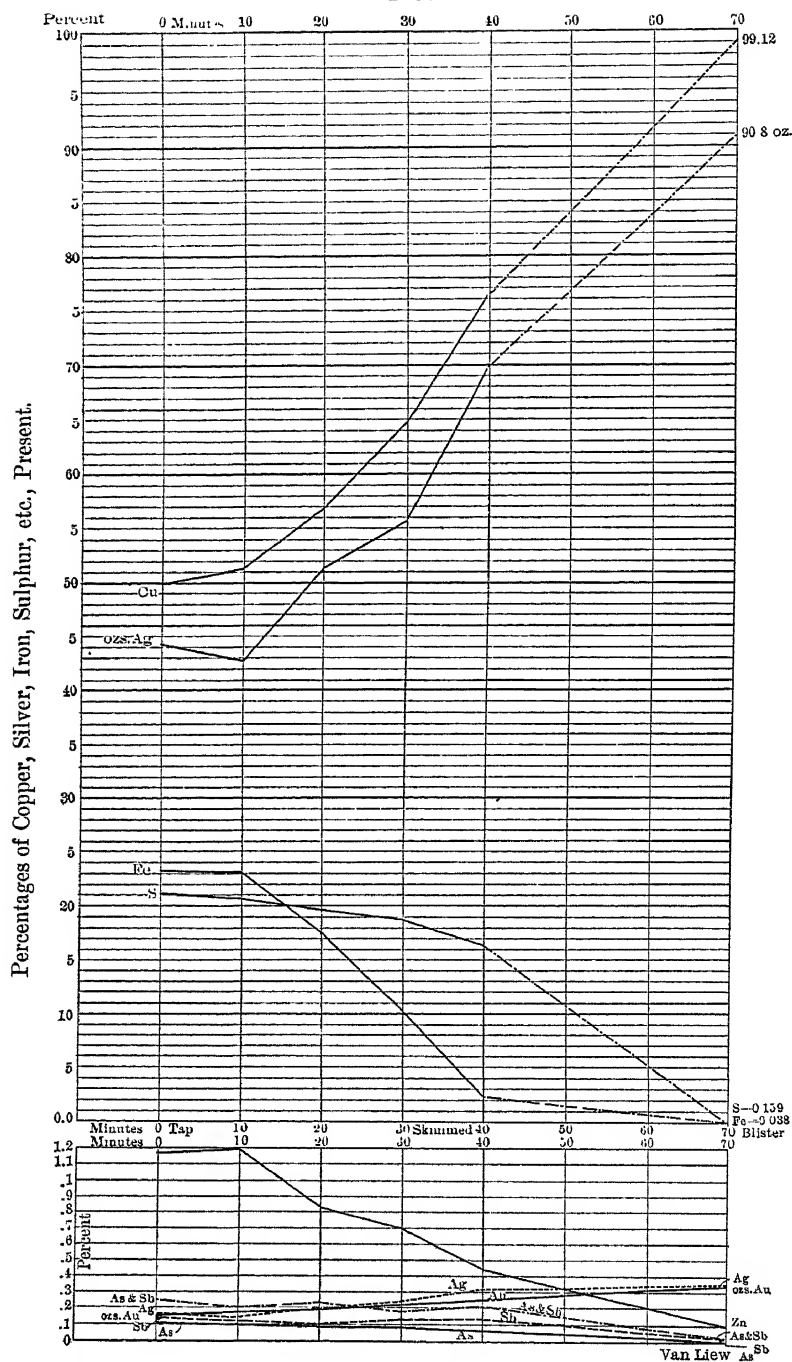
TIME.	Cupola-Tap.	10 Min.	20 Min.	30 Min.	40 Min. Last Skim.	70 Min. Blister-Copper.
Copper, per cent.....	49.72	50.20	56.88	64.60	76.37	99.120
Iron, " .....	23.31	23.15	17.85	10.50	2.40	0.038
Sulphur, " .....	21.28	20.95	19.74	18.83	16.30	0.159
Zinc, " .....	1.19	1.20	0.84	0.70	0.45	0.090
Arsenic, " .....	0.11	0.09	0.08	0.08	0.08	0.0012
Antimony, " .....	0.14	0.12	0.10	0.13	0.13	0.006
Silver, ounces.....	44.20	42.90	51.40	55.80	70.00	90.800
Gold, " .....	0.16	0.14	0.20	0.24	0.32	0.350

Fig. 1 illustrates graphically the history of their elimination. In this figure, the line of abscissas represents the ten-minute periods of blowing, and the ordinates represent percentages of copper, etc., from 0 to 100. The ordinates of the figure at the bottom are the same, but on a larger scale (from 0 to 1.2 per cent.), to show better the impurities in the matte carried to the extent of but a few tenths of 1 per cent. The upper diagram shows that the silver almost parallels the enrichment of the matte in copper.

The lines showing the relative elimination of the iron and the sulphur are the most interesting. For the first ten minutes of the blow, and while the matte is heating up, the iron- and sulphur-lines are parallel. From this point there is a marked



FIG. 1.



ELIMINATION OF IMPURITIES IN BESSEMERIZING COPPER-MATTE.

change; the sulphur-line is very gradual in its drop, showing that but little is being burned in comparison with what is taking place with the iron, whose line takes a sudden drop. The iron decreases during thirty minutes from 23.15 per cent. to 2.4 per cent. at the skimming-point, while at this point there still remains 16.3 per cent. of sulphur in the matte. From this point, however, to blister-copper, it is the sulphur that bears the brunt of elimination, the iron dropping only from 2.4 per cent. to 0.0038 at blister-copper, while the sulphur decreases from 16.3 per cent. to 0.15 per cent.

This is of great interest, as it shows that, up to the skimming-point, it is the oxidation of the iron to ferrous oxide, and the union of the ferrous oxide with the silica of the lining, that affords the source of heat to carry on the operations within a converter; while from the skimming-point (76.4 per cent. copper) to the finished blister-copper it is chiefly the burning of the sulphur that gives our heat-supply to finish the work started by the oxidation of iron.

The zinc, it will be seen, is scarcely affected during the "heating-up" period; while after that its elimination is gradual. The arsenic and antimony, curiously enough, are but slightly affected during the whole of the slag-forming period, or as long as enough iron remains to be slagged off. At the cupola-tap of matte into the converter, the arsenic was 0.11 per cent. and the antimony 0.14 per cent., while at the end of the slag-forming period the arsenic was 0.08 per cent. and the antimony 0.13 per cent.

When the iron in the matte had been oxidized and slagged off, and then only, did the arsenic and antimony commence to be oxidized and driven off, until, at the point of blister-copper, but 0.0012 per cent. of arsenic and 0.006 per cent. of antimony remained.

# Hot-Blast Smelting for the Elimination of Arsenic, Antimony, Lead and Zinc from Copper-Mattes, and for the Production of Lead.

BY S. E. BRETHERTON, VAL VERDE, ARIZONA.

(New York Meeting, October, 1903.)

MR. ALLAN GIBB, of Mount Perry, Queensland, Australia, in an interesting and instructive paper,\* describes fully the great difficulties metallurgists encounter in seeking to produce marketable copper by smelting refractory ores containing arsenic, antimony and bismuth. After reading this description, I prepared a paper for the Institute, which was, however, through a misunderstanding on my part, published in the *San Francisco Mining and Scientific Press* of April 4, 1903, after its acceptance by the Council, and without credit to the Institute. Consequently, it cannot be reprinted for the *Transactions* of the Institute; and I can only here summarize its contents and conclusions, referring the reader to the article above named for further particulars.

In that article I omitted the consideration of bismuth, which we do not encounter in our smelting at Val Verde, and added lead and zinc, which Mr. Gibb had not discussed; and I undertook to show, by the record of a year's practice of the Val Verde Copper Co., Arizona, that copper-ores containing arsenic, antimony, lead and zinc may be successfully smelted, without preliminary roasting, in an ordinary blast-furnace with hot-blast, making a normal slag, such as one would aim to make under ordinary conditions with cold-blast, so as to produce a marketable matte, containing not less than 45 per cent. of copper, free from arsenic and antimony, and below the limit in lead and zinc penalized by the refineries.

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\* "The Elimination of Arsenic, Antimony and Bismuth from Copper," *Trans.*, xxxiii., 653.

1. *Hot-Blast Copper Matte-Smelting.*

Our ore-mixture (smelting in 1200-lb. charges) had in the winter of 1902-3 the following composition:

*Analysis of Ore-Charge.*

	Per Cent.
Cu (wet), . . . . .	2.5
Fe, . . . . .	23.0
MgO and CaO, . . . . .	3.5
S, . . . . .	21.5
As, . . . . .	7.1
SiO <sub>2</sub> , . . . . .	21.6
Al <sub>2</sub> O <sub>3</sub> , . . . . .	3.7
Zn, . . . . .	4.5
	<hr/> 92.4

The remaining 7.6 per cent. was CO<sub>2</sub>, O, Pb, etc. We obtain all our iron from iron sulphides. In addition to the 1200 lb., 300 lb. of coarse slag was used on account of the fine ore (sulphides), making a total charge of 1500 lb., with which we used 60 lb. of coke, containing 25 per cent. of ash, and blast heated from 400° to 450° F. Raising the blast-temperature did not practically save fuel. Our first matte was rich in gold, and averaged from 23 to 30 per cent. of copper, no arsenic or antimony, with a trace of lead and zinc. This matte was resmelted once with siliceous copper-ores, sometimes partly oxidized, to produce the from 45- to 50-per cent. matte for shipping. The siliceous ores were then, as now, any class obtainable in the market, assaying from less than 0.5 up to 10 per cent. of copper.

Since 1902, we have had less arsenic, but more sulphur, lead and zinc. We do not attempt to save the lead from ores carrying less than 10 per cent., as it is cheaper (even if we could save all the lead) to burn it off in the copper-matting furnace than to roast it, and then smelt in the lead-furnace with a high percentage of expensive coke. Ores and concentrates containing more than 10 per cent. of lead are smelted in the lead-furnace.

Our present smelting mixture is, and has been for the past three months, as follows:

	Lb.
Sulphide ore, . . . . .	320
Lime rock, . . . . .	60 to 100
Bricked sulphide concentrates, . . . . .	860 to 840
Coarse slag, . . . . .	400
Total, . . . . .	<hr/> 1640

with which we use 50 lb. of coke containing 18 to 19 per cent. of ash. At times we drop the coke to 40 or 45 lb.

The 1640 lb. of ore and flux averages: lead, 7; zinc, 7.1; copper (wet), 2.1; sulphur, 23.2; and arsenic, 1.3 per cent. The matte in this case is again free from arsenic, but only contains from 13 to 15 per cent. of copper. The lead and zinc are also over the 3 per cent. limit, but are brought below that percentage when re-run for shipping. I attribute the poorer concentration now, not so much to the slight excess of sulphur over the first charge as to the shortage of silica and the amount of lead and zinc, which are so difficult to desulphurize in ordinary roasters.

The slag has always been very clean, showing little more than a trace of gold, silver and copper.

The wood consumed for heating 3700 cu. ft. of air per min. to 400° F. amounts to from 1.5 to 2 cords in 24 hours.

This system of smelting requires much more care on the part of the feeder than one which uses plenty of fuel with roasted and oxidized ores. For, since we depend entirely on the iron in the sulphides for fluxing purposes, apart from the 9 or 10 per cent. of lime, excess of fuel prevents concentration by causing an increased matte-flow which carries the iron, intended for the slag, into the matte. This often causes trouble, because the slag becomes too acid and an accumulation of zinc, arsenic and lead forms in the crucible, gummy in nature, and similar in composition and appearance to the "sows" formed in lead-smelting crucibles.

It is self-evident that the cold-blast smelting of such refractory ores as we have here, would be out of the question, not only on account of the extra expense for roasting and extra coke in the blast-furnace, but because arsenic and lead would be reduced and retained by the matte, so as to make it unmarketable.

We have a simple arrangement for heating the blast, designed and patented by the writer, which we do not find very expensive. The principle is that of a tubular boiler.

## 2. *Hot-Blast Lead-Smelting.\**

We have made a short run with hot-blast upon lead-sulphide, containing a very little, partially oxidized, iron; but the test was not long enough to decide the question of the value of a hot-blast in smelting lead-ores, whether as to saving of fuel, regular running of furnace, loss of metal or general technical efficiency or economy. Nevertheless, it has confirmed my impression that *warm* blast can be employed in the lead blast-furnace with considerable resultant saving of fuel, if care be taken to secure the reduction of the lead without too much loss by oxidation; and before we resort to extensive roasting and cold-blast smelting (in a place where all fuel is so dear as it is in Val Verde), we shall certainly give the warm blast another fair trial.

## 3. *General Conclusions.*

For the past three years, I have had to contend with my metallurgical colleagues, who hold that, to be of any benefit by reason of its temperature, the blast must be heated to 600° F. or more. Admitting the theoretic advantage of the higher temperature, I still maintain that 500° F. is the limit for technical economy in pyritic smelting; and the amount of fuel that can be saved by using blast at only 200° or 300° F. is surprising. In general practice, I believe 16 per cent. of coke would be used with cold-blast, and (since a part of the sulphur in the charge is utilized even with cold-blast), under favorable conditions, the amount of coke needed for that system of smelting may be as low as 12 per cent. of the charge. But we have been running regularly here for months on 5 per cent. of coke, and sometimes, for several days at a time, with only 3 per cent.

At present (June, 1903) we are running one rectangular copper-matte smelting-furnace, and doing excellent work on material containing very little copper; our blast for this furnace being heated with the hot-blast stove originally constructed to do this work for the small round furnace which

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\* SECRETARY'S NOTE.—This, and the succeeding, portions of Mr. Bretherton's paper were written in June, 1903, and have not been published elsewhere.—  
E. W. R.

gave the results cited in the first part of this paper. The charge (including 400 lb. of slag) is 2540 lb., containing 25 lb. of copper. The product is matte containing from 20.5 to 23 per cent. of copper. The furnace works perfectly; the only trouble being that the men employed as feeders have hard work to keep up with it. Our raw concentrates are moulded into briquettes in a Chisholm, Boyd & White press, and these briquettes are used to flux sulphide-ores carrying silica in excess.

The general propositions, as to the use of heated blast in the treatment of sulphides in the blast-furnace, seem to be :

1. That the heat required for effecting what may be called the "net" reduction of the raw material of the charge into metal or matte, and for effecting and maintaining the fusion of the metal, matte, slag, etc., may be furnished by the fuel actually put into the furnace, and burned by the blast, or by the pre-heating of the blast, or by the oxidation of sulphur, etc., in the charge.

2. That of these sources of effective heat, the first two cost money, while the third does not.

3. That the third can be more effectively utilized if the second also be called into play (that is to say, if the blast be pre-heated).

4. That the economic limits and advisability of this measure of economy in smelting must be determined in each case after due consideration of all the conditions, including not only the difficulty and cost of securing a suitable ore-mixture, but also the local cost of coke and other materials involved in the problem.

5. That, for many localities in the United States, the use of a warm blast in matte-smelting or lead-smelting is worth considering.

## An Example of the Alteration of Fire-Brick by Furnace-Gases.

BY FRANK FIRMSTONE, EASTON, PA.

(New York Meeting, October, 1903.)

THE furnace from which the brick here referred to were taken, was lined under my supervision and blown-in in May, 1902. It was 75 ft. high and 18 ft. in greatest diameter, and used coke to smelt a lean magnetic iron-ore.

The lining was put in as follows:—First, blocks 14 in. long by 6 in. thick; second, 2 in. of loam; third, 13.5 in. of 9-in. fire-brick (one-and-a-half brick wall) laid in English bond; fourth, red brick to within 2 in. of the iron shell; and fifth, 2 in. of slag-wool between the red brick and the shell.

The brand of brick used has been known to me for over 20 years, and in my observation had always proved very durable, notably when used in a small furnace 50 ft. high which treated the same magnetic ore. On revisiting and examining the furnace in March, 1903, it was found that the whole of the lining of 14-in. blocks at the top, and for an unknown distance (more than 20 ft.) below, had completely disappeared; and a few weeks later the 13.5-in. lining and the red brick at the top likewise vanished, and the furnace was blown out.

Notwithstanding the complete failure of the lining higher up, measurements taken after blowing-out showed no unusual wear in the lower part of the furnace.

The exact position in the furnace of the piece of brick-work used in the analysis, given below, is not known; but it came from between 30 and 40 ft. above the bottom. It had formed part of the 13.5-in. wall (no 14-in. brick remaining at that level) and consisted of parts of several bricks so firmly sintered together that fracture occurred indifferently through the joints and through the body of the brick. The joints could, however, be traced by a close examination. The surface which had been in contact with the material in the furnace was covered with a yellowish, translucent glaze, somewhat vesicular, and from



$\frac{1}{8}$  in. to  $\frac{1}{2}$  in. thick. The body of the brick was much stronger than a new one; the fracture resembled porcelain of bluish-grey color with occasional spots (near the glazed surface) nearly black. No deposited carbon was visible, nor was any carbon noticed in the course of the analysis, although it was looked for.

*Analyses of Parts of the Furnace-Lining.*

By Dr. P. W. Shimer.

Components.	Glazed Skin About $\frac{1}{8}$ In. Thick	Body of Brick.	Joint. (b)
	Per Cent.	Per Cent.	Per Cent.
Silica.....	40.23	57.63	.....
Alumina (a).....	12.22	31.64	.....
Iron protoxide.....	11.93	3.73	6.60
Lime.....	10.92	0.24	.....
Magnesia.....	4.31	0.11	.....
Potash.....	9.39	2.59	2.39
Soda.....	8.43	0.51	0.79
Metallic iron.....	.....	1.51	.....
Zinc oxide.....	.....	0.59	.....
Total.....	97.43	98.55	.....
Total iron.....	9.28	4.41	4.62

(a) Includes any titanium oxide present.

(b) The bricks were laid in a mortar of ground clay and crushed brick, both furnished by the makers.

The sample analyzed from the body of the brick was taken from a point at least an inch away from the glazed surface and at least that distance from any joint.

*Analyses of Brick from the Same Makers, before Use.*

By Dr. P. W. Shimer.

Components.	9-in. Brick Used in Lining, 1902.	Brick Made in 1896.
	Per Cent.	Per Cent.
Silica.....	55.62	56.12
Alumina (a).....	38.55	37.43
Iron sesquioxide...	4.17	4.43
Lime.....	0.24	0.36
Magnesia.....	0.24	0.29
Potash.....	0.95	0.99
Soda.....	0.29	0.23
Total.....	100.06	99.90
Total iron.....	3.10	2.92

(a) Includes any titanium oxide present.

Many spots of iron oxide were visible on the fractured surface of the brick. The burnt-clay basis (chamotte) was nearly white and not very fine; the cementing clay was of light-brown or buff color.

Iron oxide and the alkalis have increased each about 50 per cent. in the altered brick, and there is a slight apparent increase in silica and a considerable decrease in alumina.

The presence of metallic iron was proved by first tritulating in an agate mortar, then treating the magnetic part of the powder with copper sulphate and again tritulating, which revealed abundant scales of metallic copper. Iron was then easily detected in globules, plainly visible with a pocket-lens on a ground surface of brick. Occasionally, these globules were large enough to be seen easily without the aid of a glass. A single hexagonal crystal (probably of zinc oxide) was seen in one cavity, but it was lost in trying to extract it.

All attempts to prepare a thin section have failed. The body of the altered brick is very hard and quite opaque, and the numerous cavities cause the section to break up long before it is thin enough to transmit a sufficient quantity of light.

The destructive action of blast-furnace gases on the lining-brick has often been noticed, and a number of papers have appeared on the subject since 1875; several of late years in *Stahl und Eisen*.

One case, occurring at a furnace in the Cleveland district in 1875, was very thoroughly investigated by Mr. John Pattinson, of Newcastle-upon-Tyne, and the results were published in *Jour. I. and S. Inst.*, No. 1, 1876, p. 85. Another, at Friederich-Wilhelmshütte, Mülheim a. d. Ruhr, was published by Limbor in *Wochenschrift des Ver. deut. Ingen.*, 1878, p. 259, and more fully, including a discussion of Pattinson's results, in *Bull. de l'Industrie Minérale*, 2d ser., tome x. (1881), p. 483.

In both of these cases the bricks in an intermediate zone of the furnace were disintegrated by carbon-deposition, brought about by the action of carbonic oxide on the spots of iron oxide in the brick, in accordance with the well-known action between carbonic oxide, iron oxide and metallic iron at certain temperatures; while lower down, the bricks were largely impregnated with alkaline salts. In the case I am now report-

ing, I have no doubt that the same action destroyed the brick in the upper part of the furnace, but positive evidence is lacking because of the total disappearance of the bricks a little above the place from which the piece analyzed was taken. In confirmation of this opinion it was noticed that the cast-iron hopper had been much attacked by the gas, and presented, in places, considerable cavities filled with carbon and particles of disintegrated cast-iron. There were also considerable deposits of carbon in several places where two castings had touched without making a gas-tight joint.

This action of the gas on cast-iron has been less noticed than that on fire-brick. Before the general substitution of closed for open-topped furnaces in the Lehigh Valley (before 1870), it was a common practice to protect the bricks near the top of the furnace against mechanical wear by the use of cast angle-irons, one leg being laid in the bed-joint of the bricks and the other covering the face.

After the furnace-tops were closed, it happened, in several cases, that so great a carbon-deposition in the bed-joints was provoked by the presence of these angle-irons, that the lining-rings and hoppers were raised as much as 9 or 10 in., and had to be reset.

Not only was there much carbon-deposition in the joints, but the castings themselves were disintegrated in places, and swelled to nearly double their original thickness. Specimens of such castings in my possession, taken from No. 2 Furnace at Glendon, in 1878, show that this action was very irregular, without clearly revealing why some spots were attacked while the rest of the casting escaped. In some of them, however, the action had plainly begun where the gates had been knocked off (which left a surface not covered by the skin of the casting), and from the gates it had extended under the skin which had been raised up and even curled over in a remarkable manner.

These difficulties caused the general abandonment of the angle-plates, and at Glendon we substituted cast-iron aprons hung by bolts from the lining-rings. In exceptional cases these also were attacked in spots, but only in furnaces making foundry iron and, consequently, running on a relatively light burden.

Mr. Pattinson points out that failures such as he has reported must be due to exceptional circumstances, and the same may be said in the present case. It is true that the bricks are higher in iron than is desirable, but exactly similar bricks (to judge by the analysis and by the appearance of the fractured brick) have never before, in my observation, failed in like manner.

In Mr. Pattinson's case it is probable that the furnace had been more or less scaffolded, perhaps from the use of fine ore (*op. cit.*, p. 92); at the Friederich-Wilhelmshütte, the furnace had been making foundry-iron. In the case under discussion, owing to the state of the coke-supply in the summer of 1902, the furnace was several times stopped for want of coke between the beginning of August and the middle of November, standing at times for several weeks. This probably caused more or less scaffolding, and certainly caused the furnace to be run frequently with a very light burden.

It is easy to see that scaffolding may favor carbon-deposition; also, that the local rise in temperature, which is sooner or later manifest over a scaffold (where the cooling effect of the descending materials is diminished or totally suspended) may be favorable to the impregnation of the brick by alkaline vapors.

Unlike Pattinson's and Limbor's cases, this case showed no sign of soluble salts in the brick analyzed. The discovery of iron carbonyl by Messrs. Mond and Quincke,\* suggests a possible, but not probable, means, whereby the increased percentage of iron may have been introduced into the brick.

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\* *Jour. Soc. Chem. Ind.*, 1891, p. 644.

## The Assay of Zinc-Box Residues from the Cyanide Process.

BY ASST. PROF. RICHARD W. LODGE, OF MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, October, 1903.)

SEVERAL methods, both wet and dry, for the assay of zinc-box residues from the cyanide process, have been described in recent years, and each of them has been claimed to be superior to all others. In the year 1901, a paper, entitled "Assay of Zincy Precipitates," was published in the *School of Mines Quarterly* to the purport that the scorification method for the assay of zinc-box residues was absolutely unreliable.

In order to shed light on this matter, the following experiments were undertaken by Messrs. C. B. Hollis and F. D. Kehew, undergraduate students at the Massachusetts Institute of Technology.

The zinc-box residues used were obtained through the courtesy of Mr. H. R. Batcheller. The samples were very rich and varied greatly in the fineness of their condition.

### SCORIFICATION-ASSAYS.

In the preliminary tests, the charge, which was weighed on a chemical balance, consisted of 0.1 Assay Ton of residues mixed with from 30 to 35 grammes of test-lead and placed in a 3-in. scorifier, over this an additional quantity of test-lead (from 30 to 35 grammes) was placed for a cover, and borax-glass, varying in quantity from 3 to 15 grammes, was sprinkled over the top of each charge of the various assays for a cover. The charges were scorified in a muffle-furnace heated to the ordinary temperature which is used in scorification; in some cases the door of the muffle was left open, while in others it remained closed. Generally, the charges spit badly, especially in the assays that were made with door of the muffle left open, or in those in which the door was opened too quickly. The results of the preliminary assays showed: 1. That in order to obtain approxi-

mately uniform results, the material submitted to the assay must be in sufficiently fine condition to pass through a 200-mesh sieve. 2. That the ordinary chemical balances are not sufficiently delicate to afford accurate results in handling these residues which are so rich in gold and silver. 3. That a large quantity of borax-glass is absolutely necessary (from 3 to 10 grammes for 0.1 Assay Ton of residues); and 4. That spitting can be avoided, provided the muffle be heated to a high temperature before the introduction of the charge, and provided the door of the muffle be kept closed until the content of the scorifier has become thoroughly liquefied; after this the temperature may be lowered.

The quantity of zinc-box residues received amounted to 458 grammes and, upon sizing, it was found that 146 grammes, or 31.8 per cent., remained upon a 125-mesh screen; 64 grammes, or 13.9 per cent., passed through a 125-mesh screen and was caught on a 160-mesh screen; and 248 grammes, or 54.1 per cent., passed through a 160-mesh screen. The entire quantity of residues was then put on a 160-mesh screen and the material that sifted through was treated on a 200-mesh bolting cloth, yielding 290 grammes of very fine material, less than 200-mesh in size, on which the tests were made.

In order to mix the sample thoroughly, the entire quantity of fine material was placed in a 38-oz. bottle closed with a glass stopper and steadily shaken for 20 minutes, the bottle and its content being alternately shaken and rotated. The mixed material was then poured out upon a glazed paper, on which it was rolled 100 times, finally being spread out in a thin layer covering an area 18 in. square. Spatula-samples to the number of 450 were then taken, which constituted a new sample, weighing 102 grammes. A chemical analysis of the new sample showed that it contained 9.09 per cent. of copper and 14.3 per cent. of zinc.

The assays were made in a muffle-furnace heated with coke, and the cupels used were of the ordinary bone-ash variety made at the Institute of Technology. Ninety per cent. of the material forming the cupels was of sufficient fineness to pass through an 80-mesh screen.

## SCORIFICATION-ASSAY.

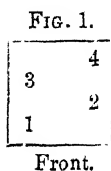
Four charges, Nos. 1, 2, 3 and 4, each of 0.05 Assay Ton in weight, were weighed on an assay balance sensitive to 0.02 of a milligramme and treated as follows:—

No. 1. 0.05 Assay Ton of the residues was mixed with 35 grammes of test-lead in a 3-in. scorifier; 30 grammes of test-lead were then added to the top of the charge, followed by a cover of 10 grammes of borax-glass.

No. 2. The same as No. 1.

No. 3. 0.05 Assay Ton of the residues was mixed with 6 grammes of litharge in a 3-in. scorifier. Additional test-lead was added, amounting to 40 grammes, followed by a final cover of 10 grammes of borax-glass.

No. 4. 0.05 Assay Ton of the residues was mixed with 1 gramme of fine charcoal and 35 grammes of test-lead; the mixture was then covered with 30 grammes of test-lead and a final cover of 10 grammes of borax-glass. The scorifiers were placed in the muffle as shown in Fig. 1. The muffle was very hot and the door was kept closed for about 5 minutes, after which it was opened.

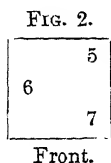


Charge No. 1 spit badly, doubtless due to its position in the muffle; charge No. 3 spit to a slight extent; while charges Nos. 2 and 4 did not spit. Charge No. 4 became covered over very quickly, owing to the charcoal in the mixture, but the resultant button of lead was so large that it was necessary to rescorify it. Although zinc-ores require a high temperature for the fusion, the heat was lowered as soon as possible after the muffle was opened, in order to slag off the copper and avoid a second scorification. The fused material poured well and the color of the scorifier indicated that the buttons could be cupelled with safety. Both the slags and the cupels were assayed by the crucible method, the results being given in Table I. The silver- and gold-beads from the slag of charges Nos. 1 and 2, and those from the cupels used in tests Nos. 2 and 3, sank into the cupels which were reassayed. This additional assay may account for the low results in silver and gold that were obtained in tests Nos. 2 and 3.

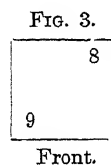
The silver-gold beads were weighed, but as they did not

contain sufficient silver to part them they were recuperated, with the addition of chemically pure silver; the cupels of this latter cupellation were not reassayed. The parting was done with nitric acid of 1.16, 1.2 and 1.27 specific gravities. The results obtained are given in Table I.

Charges Nos. 5, 6 and 7, which were similar in all respects to charges Nos. 1 and 2, were next weighed and placed in the muffle as indicated in Fig. 2. The temperature of the muffle was that used in the ordinary assay, and the door of the muffle was kept closed for 10 minutes. It was then opened and charge No. 6 was seen to spit twice; the door was then closed and, through an opening in the muffle, charge No. 6 was seen to spit a third time. The door of the muffle was then opened and the scorification completed. The slags and the cupels from these charges were assayed as in the former tests, the results being given in Table I.



In order to ascertain whether a scorifier of larger size would be beneficial or not, charge No. 8 was assayed in a 3-in. scorifier, and charge No. 9 in a 4-in. scorifier. Charge No. 8 consisted of 0.05 Assay Ton of residues placed in the bottom of a 3-in. scorifier and covered with 65 grammes of test-lead, followed with a final cover of 10 grammes of borax-glass. Charge No. 9 consisted of 0.05 Assay Ton of residues, prepared as in charges Nos. 1, 2, 5, 6 and 7, with the exception that a 4-in. scorifier was used in place of a 3-in. one. The tests were placed in the muffle as shown in Fig. 3. The muffle was closed and both charges were seen to spit badly. They were then allowed to become covered and were later poured and treated in a manner similar to the earlier tests. From the results of these tests, which are given in Table I., charges Nos. 8 and 9 were rather peculiar. The gold in No. 9 was very low, the silver was very high and an exceedingly large quantity of silver was recovered from the slag. Charge No. 8, also, showed a high percentage of silver in the slag. The cause of these odd results was not apparent.



Charges Nos. 10, 11 and 12 were then made as follows:—

Charge No. 10 consisted of 0.05 Assay Ton of residues, 60 grammes of test-lead, 10 grammes of litharge and 12 grammes of borax-glass, thoroughly mixed together in a 4-in. scorifier.



TABLE I.—Data of Scorification-Assays of Zinc-Bx Residues Containing Copper, 9.09 Per Cent., and Zinc, 14.3 Per Cent. Mr. Hollis.

Number of Charge.	Weight of Gold						Weight of Silver.						Grand Totals.					
	Grammes in First Button.	Grammes in Slag.	Per Cent.	Grammes in Cupel.	Per Cent.	Un-ces Per Ton.	Grammes in First Button.	Grammes in Slag.	Per Cent.	Grammes in Cupel.	Per Cent.	Total.	Un-ces Per Ton.	Gold and Silver Ounces.	Per Cent Loss of Gold.	Per Cent Loss of Silver.	Per Cent Loss in Gold and Silver.	Number of Charge.
1	0.23426	0.00010	0.04	0.00022	0.09	0.23458	0.20564	0.00148	0.70	0.00282	1.34	0.20091	4.108.8	8800.4	0.13	2.01	1.01	1
2	0.23368	0.00012	0.05	0.00018	0.08	0.23418	0.20436	0.00128	0.66	0.00208	1.09	0.20782	4.156.4	8810.0	0.13	1.06	0.85	2
3	0.23146	0.00026	0.11	0.00024	0.10	0.23196	0.20308	0.00172	0.83	0.00262	1.26	0.20742	4.148.4	8872.6	0.21	2.09	1.09	3
4	0.23464	0.00018	0.08	0.00018	0.08	0.23472	0.20320	0.00236	1.28	0.00306	1.46	0.20692	4.178.4	8872.4	0.16	2.74	1.37	4
5	0.23422	0.00022	0.09	0.00028	0.12	0.23470	0.20388	0.00310	1.47	0.00300	1.42	0.21048	4.209.6	8869.2	0.21	2.40	1.11	5
6	0.23418	0.00024	0.10	0.00028	0.12	0.23470	0.20372	0.00158	0.75	0.00346	1.65	0.20576	4.175.2	8879.2	0.22	2.40	1.15	6
7	0.23436	0.00012	0.05	0.00044	0.19	0.23492	0.20440	0.00140	0.67	0.00321	1.55	0.20604	4.180.8	8879.2	0.24	2.22	1.17	7
8	0.23404	0.00018	0.08	0.00028	0.12	0.23460	0.20376	0.00316	2.89	0.00340	1.59	0.21332	4.266.4	8956.4	0.20	4.48	2.91	8
9	0.23222	0.00020	0.09	0.00020	0.09	0.23262	0.20356	0.02718	11.63	0.00296	1.26	0.23370	4.674.0	9326.4	0.18	12.89	6.55	9
10	0.23222	0.00030	0.13	0.00034	0.15	0.23286	0.20228	0.00396	2.82	0.00276	1.31	0.21100	4.220.0	9317.2	0.28	4.13	2.12	10
11	0.23222	0.00106	0.45	0.00080	0.13	0.23368	0.20400	0.00254	1.21	0.00288	1.36	0.20910	4.168.0	8859.6	0.63	2.57	1.52	11
12	0.23260	0.00032	0.14	0.00024	0.10	0.23316	0.20362	0.00288	1.38	0.00248	1.18	0.20808	4.179.6	8842.8	0.54	2.56	1.34	12
13	0.23360	0.00048	0.20	0.00016	0.20	0.23444	0.20331	0.00201	1.38	0.00366	1.74	0.21018	4.203.6	8892.4	0.20	3.12	1.69	13
14	0.23344	0.00070	0.20	0.00016	0.20	0.23414	0.20415	0.00181	....	....	....	0.20696	4.119.2	8802.0	....	....	....	14

TABLE II.—Data of Assays of Slags and Cupels Obtained in Treating Zinc-Bor Residues of Table I.

First Scorification.				Second Scorification.				Slag Charge.				Charge for Fusing the Cupels.														
No. of Charge.	Weight of Zinc Residue.	Weight of Pb.	Weight of Bo- rax-Glass.	Time in Min.	Weight of Pb. Button.	Time Cupelling.	Weight of Slag.	Soda.	PbO.	Argols.	Glass.	Time of Fusion.	Grammes.	Weight of Button.	Time of Fusion.	Glass.	Argols.	PbO.	Borax-Glass.	Weight of Cupel.	Soda.	Grammes.	Minutes.	Time of Fusion.	Weight of Button.	No. of Charge
Grammes.				Grammes.				Grammes.				Grammes.				Grammes.				Grammes.						
1	$\frac{1}{4}$ A. T.	65	10	30	21	20	59	15	30	2	10	25	28	Not taken.	35	15	2	60	None.	15	Not taken.	15	20	35	20	1
2	$\frac{1}{4}$ A. T.	65	10	25	22	21	59	15	30	2	10	25	24	Not taken.	35	15	2	60	None.	15	Not taken.	15	17	35	17	2
3	$\frac{1}{4}$ A. T.	40 Pb. 6 PbO.	10	30	13	15	49	15	30	2	10	25	25	Not taken.	35	15	2	60	None.	15	Not taken.	15	22	35	22	3
4	$\frac{1}{4}$ A. T.	1 gramme charcoal	10	28	32	20	114	15	30	2	10	25	20	Not taken.	35	15	2	60	None.	15	Not taken.	15	20	35	20	4
5	$\frac{1}{4}$ A. T.	65 Pb.	10	33	36	23	54	15	30	2	10	25	25	25	42	35	15	2	60	5	42	Not taken.	15	28	35	5
6	$\frac{1}{4}$ A. T.	65 Pb.	10	33	21	19	54	15	30	2	10	25	24	30	35	15	2	60	5	30	Not taken.	15	23	35	23	6
7	$\frac{1}{4}$ A. T.	65 Pb.	10	40	15	15	64	15	30	2	10	25	24	49	35	15	2	60	5	49	Not taken.	15	21	35	21	7
8	$\frac{1}{4}$ A. T.	65 Pb.	10	28	26	25	54	15	30	2	10	25	25	53	35	15	2	60	10	53	Not taken.	15	20	35	20	8
9	$\frac{1}{4}$ A. T.	65 Pb.	10	43	11	18	72	15	30	2	10	25	20	25	35	15	2	60	10	25	Not taken.	15	20	35	20	9
10	$\frac{1}{4}$ A. T.	60 Pb. 10 PbO.	12	28	15	19	69	15	30	2	10	25	21	31	35	15	2	60	10	31	Not taken.	15	20	35	20	10
11	$\frac{1}{4}$ A. T.	60 Pb. 10 PbO.	12	29	15	18	75	15	30	2	10	25	20	80	35	15	2	60	10	80	Not taken.	15	28	35	28	11
12	$\frac{1}{4}$ A. T.	65 Pb.	14	25	20	23	65	15	30	2	10	25	20	39	35	15	2	60	10	39	Not taken.	15	24	35	24	12
13	$\frac{1}{4}$ A. T.	60 Pb. 10 PbO.	12	35	16	?	73	15	30	2	10	25	20	37	35	15	2	60	10	37	Not taken.	15	22	35	22	13
14	$\frac{1}{4}$ A. T.	60 Pb. 10 PbO.	12	40	10	14	76	15	30	2	10	25	19	20	35	15	2	60	10	20	Not taken.	15	21	35	21	14

Charge No. 11 consisted of 0.05 Assay Ton of residues mixed with 30 grammes of test-lead; on this were placed an additional 30 grammes of test-lead, followed by a cover of 10 grammes of litharge and a final cover of 12 grammes of borax-glass.

Charge No. 12 consisted of 0.05 Assay Ton of residues mixed with 30 grammes of test-lead in a 4-in. scorifier, over which were placed 30 grammes of test-lead and a final cover of 14 grammes of borax-glass. Charges Nos. 10, 11 and 12 were placed in the muffle as shown in Fig. 4, and the door of the muffle was kept closed for 10 minutes. Charge No. 10 fused very quietly and did not even tend to jump. Charge No. 11 was less quiet, but did not spit, and charge No 12 was quiet. The charges were allowed to become covered and were then poured, the resultant buttons, slags and cupels being assayed as in the former tests. (See Table I.)

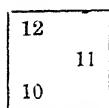
The results from charges Nos. 11 and 12 were low, and the gold obtained from charge No. 10 was especially so. The silver-gold button obtained from the assay of the cupel used for charge No. 10 sank into the cupel, which had to be reassayed.

Charge No. 13 was similar to charge No. 10, and consisted of 0.05 Assay Ton of residues, 60 grammes of test-lead, 10 grammes of litharge and 12 grammes of borax-glass, all thoroughly mixed together in a 4-in. scorifier.

Charge No. 14 consisted of 0.05 Assay Ton of residues and 30 grammes of litharge, mixed together in a 4-in. scorifier, and covered with 30 grammes of test-lead, with a final cover of 12 grammes of borax-glass.

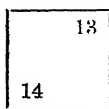
Charges Nos. 13 and 14 were placed in a hot muffle in the position shown in Fig. 5. The muffle was then closed for 10 minutes. Charge No. 13 fused quietly and had no tendency to spit, while charge No. 14 spit several times after the door of the muffle was opened. These charges were treated in a manner similar to the previous tests, except that, in the assay of the cupel of test No. 14, trouble was encountered with the silver-gold button, which accounts for lack of results given under this heading in Table I. The results for gold in tests Nos. 13 and 14 were low, an effect which seems to be true of all assay-charges containing litharge.

Fig. 4.



Front.

Fig. 5.



Front.

A complete summary of the data obtained in tests Nos. 1 to 14, inclusive, are given in Table I.; and in Table II. are given the weights of the lead buttons and other data relative to the assays of the slags and cupels of these tests.

A study of the results shows that the addition of charcoal to the charge seems to aid the scorification. Also, that all charges in which litharge was used (Nos. 3, 10, 11, 13 and 14) gave low results for gold. The addition of the litharge, however, seemed to prevent the spitting of the charge during fusion.

The value of the residues in gold lies evidently between 4,690 and 4,698.4 oz. per ton, as the results from 6 of the 14 charges are within these limits, and 3 of these 6 are practically identical, i.e., 4,694 oz. The results for silver in 4 of the tests were between 4,175.2 and 4,179.6 oz. per ton; 4,178.5 oz. per ton being taken for the quantity present.

Estimating the value of gold at \$20.67 per oz. and silver at \$0.50 per oz., the value per ton of the residues was:—for gold, \$97,025, and for silver, \$2,089. On the basis of these values, a comparison of the highest and lowest content of gold as determined by the scorification method and the percentage of variation from the correct value is:—

Ounces above, 4.4, value \$91, or 0.09 per cent.

Ounces below, 96.8, value \$2,000.85, or 2.06 per cent.

The extremely low results of test No. 10 (54.8 oz., valued at \$1,132.71, or 1.16 per cent.) are omitted from this calculation.

Summarizing the results of the determinations of silver in a similar manner, there were obtained the following figures:—

Ounces above, 495.5, value \$247.7, or 11.85 per cent.

Ounces below, 30.1, value \$15, or 0.72 per cent. The high results obtained in test No. 9 (87.9 oz., valued at \$43.95, or 2.1 per cent.) were not included in this calculation.

The results for the silver determinations were less uniform than were those for the gold, but the value is apparently between 4,175 and 4,220 oz. per ton, 9 results out of the 14 being within this range. The charges containing litharge gave low results for silver in 2 out of 5 cases.

The quantity of gold found in the slags was generally less than that found in the cupels—a result which is unusual.

From the results obtained in tests Nos. 1 to 14, the best

method for the treatment of these zinc-box residues is as follows:—The charge should consist of 0.05 Assay Ton of residues mixed with 35 grammes of test-lead in a 3- or a 4-in. scorifier and covered with a layer of 30 grammes of test-lead, followed by a final cover of from 10 to 12 grammes of borax-glass. The filled scorifier should be placed in a hot muffle (in order that the fusion shall occur rapidly) and the door should be closed for fully 5 minutes after the charge has been fused. During the time that the door is closed, no air whatever should enter the muffle. When the charge has become thoroughly fused, the door of the muffle should be opened and the remainder of the assay conducted in the usual manner.

#### CONFIRMATORY WET ASSAYS.

In order to confirm the results obtained in the scorification-assays, Mr. Hollis made a duplicate determination of gold and silver in the zinc-box residues by the wet method of Mr. C. Whitehead. These results (charges Nos. 15 and 16) were, respectively, for gold, 4,698.8 and 4,694.8 oz. per ton; and for silver, 3,841.6 and 3,555.6 oz. per ton. The results for gold are practically the same as those obtained in the scorification-assay. The results for the silver, however, are very much lower and are doubtless due to the incomplete precipitation of the silver bromide which is soluble to a certain extent in too strong a solution of potassium bromide. This effect is analogous to the action of silver chloride, for, if a solution of silver nitrate be precipitated by salt in a solution that is not sufficiently diluted, the silver chloride will not be all thrown down, some of it being dissolved in the strong brine.

The residues were assayed also by the wet method suggested by Messrs. Charles H. Fulton and C. H. Crawford,\* which is called the "combination wet and dry method of assay."

The method was used exactly as described, with the exception that the filter and content were not scorified, but were assayed in a glazed crucible after having been separately burned. The data obtained by the combination wet and dry method are given in Table III.

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\* *School of Mines Quarterly*, January, 1901, p. 157.

TABLE III.—Data of Assays of Zinc-Box Residues by the Combination Wet and Dry Method. Mr. Hollis.

Number of Charge.	Weight of Gold.				Weight of Silver.				Grand Total.			
	In First Button.	In Slag.	In Cupel.	Total.	Per Cent.	In First Button.	In Slag.	In Cupel.	Total.	Per Cent.	Loss in Gold.	Loss in Silver.
	Grammes.	Grammes.	Grammes.	Ounces Per Ton.		Grammes.	Grammes.	Grammes.	Ounces Per Ton.		Ounces Gold and Silver.	Per Cent.
17	0.23270	0.00068	0.00076	0.23352	0.32	0.20220	0.00697	0.47	0.20952	1.23	4.113.8	0.98
18	0.23422	0.00010	0.00001	0.23436	0.02	0.20252	0.00651	0.25	0.20300	1.12	4.106.6	0.67
19	0.23390	0.00032	0.00022	0.23444	0.09	0.20061	0.00173	0.84	0.20225	1.09	4.092.4	1.03
20	0.23408	0.00026	0.00020	0.23454	0.09	0.20144	0.00129	0.63	0.20300	0.97	4.094.6	0.85
21	0.23350	0.00012	0.00068	0.23430	0.41	0.20440	0.00071	0.34	0.20510	1.40	4.104.2	1.11

TABLE IV.—Data of Assays of Zinc-Box Residues by the Crucible Method. Mr. Hollis.

No. of Charge.	Weight of Gold.				Weight of Silver.				Grand Total.			
	In First Button.	In Slag.	In Cupel.	Total.	Per Cent.	In First Button.	In Slag.	In Cupel.	Total.	Per Cent.	Loss in Gold.	Loss in Silver.
	Grammes.	Grammes.	Grammes.	Ounce Per Ton.		Grammes.	Grammes.	Grammes.	Ounces Per Ton.		Gold and Silver.	Per Cent.
22	0.23320	0.00028	0.00028	0.23376	0.12	0.19735	0.00569	0.00192	0.20496	0.94	8.774.2	1.86
23	0.23176	0.00146	0.00034	0.23356	0.14	0.19545	0.00687	0.27	0.20262	1.28	8.768.0	2.58
24	0.23390	0.00036	0.00100	0.23526	0.42	0.19733	0.00671	0.00324	0.20728	1.56	8.850.8	2.55
25	0.23380	0.00034	0.00026	0.23440	0.11	0.20136	0.00489	0.00236	0.20861	1.33	8.850.2	1.77
26	0.23328	0.00042	0.00060	0.23430	0.26	0.20113	0.00407	0.00306	0.20825	1.47	8.851.2	1.84

The data given in Table III. show that the results for gold were somewhat lower than those obtained by the scorification-assay, while those for silver were very much lower.

#### CRUCIBLE ASSAYS.

Three portions of residues were taken, of 0.05 Assay Ton weight, and to each was added 15 grammes of soda, 10 grammes of borax-glass, 90 grammes of litharge and 2 grammes of argols; an excessive quantity of litharge was used, in order to slag the copper and the zinc. The fusion was made in a "G" crucible, which had previously been glazed with borax-glass, and each charge was fused for 35 minutes. One charge ate through the crucible, one would not pour, the third only seeming satisfactory. The results of the good test are given in Table IV., charge No. 22. Four additional charges were made, Nos. 23, 24, 25 and 26, all similar to No. 22, with the exception that the quantity of borax-glass was increased to 15 grammes in each charge. These charges worked satisfactorily in the furnace, but the results, which are given in Table IV., were not all that was hoped for.

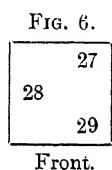
The data given in Table IV. show that the results for gold and silver averaged much lower than the quantities obtained in the scorification-assays. The quantity of silver obtained was very much lower, a result which may be due to the large quantity of litharge used in the charge, or to the quantity of copper present in the sample. The slag also from the crucible assay is richer in both silver and gold, and the second slags and second cupels should have been assayed,—an omission which is to be regretted.

In order to verify the results obtained in the scorification method by Mr. Hollis, a duplicate set of experiments were made by Mr. Kehew on residues from the same lot of samples. Mr. Kehew conducted the assays in a muffle that was fired by gas, and measured the temperature of the experiment tests by a Le Chatelier pyrometer.

The same care was observed in taking the samples, and the same button-balance was used to weigh the samples, although a different set of assay weights was used.

Three charges were made, as follows:—0.05 Assay Ton of residues was mixed with 35 grammes of test-lead in a 3-in.

scorifier; on this were placed 30 grammes of test-lead and a final cover of 10 grammes of borax-glass. The charges were placed in a very hot muffle, which was of the dimensions 12 in. by 6.25 in. by 4 in., and the door was closed for 5 minutes; the door was then opened and the heat lowered. No spitting took place. The charges were run so that they were just driving, but upon pouring it was found that the temperature had not been sufficiently high to decompose all of the charge. The results were therefore rejected. Three similar charges were then made, Nos. 27, 28 and 29, and placed in the muffle in the position shown in Fig. 6. The test was conducted as before, but at a higher temperature (780° C. by pyrometric measurement). The resultant lead buttons were too large for cupellation and they were rescorified, with the addition of 2 grammes of silica. The second lead button was cupelled, weighed, recupelled, with the addition of C.P. silver, and parted with three strengths of nitric acid, having specific gravities, respectively, of 1.16, 1.20 and 1.28. The results obtained are given in Table V.



By noting the position of these charges in the muffle, it is seen that the quantity of gold found in the slag increased with the increased temperature, *i. e.*, it was greatest in the back of the muffle and least in the front.

Owing to the fact that these assays as conducted in a 3 or 3.5-in. scorifier yielded a button of lead which was too large to cupel, the subsequent assays were allowed to become covered over with the slag, which was then poured as much as possible; the scorifier was then replaced in the muffle and the scorification continued until the "lead-eye" was of a diameter of 0.5 in.; the content of the scorifier was then poured.

Charges Nos. 30, 31, 32 and 33 were of the same composition as Nos. 27, 28 and 29, and were placed in the muffle in the position shown in Fig. 7. The slags from these assays were ground, passed through a 40-mesh sieve and assayed. The buttons from two of the assays passed into the cupel and were lost; the other two were weighed and parted. The data given in Table V. show that more gold was recovered from the slag in tests Nos. 31 and 33, which were in the back of the muffle, than in Nos. 30 and 32, which were in the front. The tem-

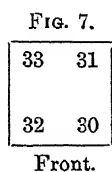




TABLE V.—Data of Scorification-Assays of Zinc-Box Residues (Duplicates of Table I). Mr. Kehew.

Gold Account.										Silver Account.						Grand Totals.				
No. of Charge.	Grammes in First Button.	Grammes in Slag.	Per Cent.	Grammes in Cupel.	Per Cent.	Total Grammes.	Ounces Per Ton.	Grammes in First Button.	Grammes in Slag.	Per Cent.	Grammes in Cupel.	Per Cent.	Total Grammes Silver.	Ounces Per Ton.	Gold and Silver.	Per Cent. Loss in Gold.	Per Cent. Loss in Silver.	Per Cent. Loss Ag. and Au.		
27	0.23404	0.00088	0.37	0.00028	0.11	0.23518	4,703.6	0.20610	0.00184	0.87	0.00218	1.03	0.21012	4,202.4	8,906.0	0.48	1.90	1.16	$\frac{1}{10}$ A. T.	
28	0.23400	0.00074	0.32	0.00028	0.12	0.23502	4,701.4	0.20614	0.00184	0.87	0.00238	1.12	0.21034	4,206.8	8,907.2	0.44	2.00	1.17	$\frac{1}{10}$ borax-glass.	
29	0.23400	0.00044	0.19	0.00022	0.09	0.23466	4,693.2	0.20648	0.00174	0.83	0.00238	1.13	0.21060	4,212.0	8,905.2	0.28	1.96	1.08	$\frac{1}{10}$ borax-glass.	
30	0.23432	0.00080	0.34	0.00016	0.07	0.23528	4,705.6	0.20314	0.00374	1.78	0.00374	1.78	0.21062	4,212.4	8,918.0	0.41	3.56	1.89	$\frac{1}{10}$ A. T.	
31	0.23446	0.00090	0.38	0.00012	0.05	0.23548	4,709.0	0.20680	0.00268	1.27	0.00432	2.03	0.21080	4,216.0	8,925.6	0.43	3.82	1.79	$\frac{1}{10}$ borax-glass.	
32	0.23420	0.00032	0.14	0.00014	0.06	0.23466	4,693.2	0.20620	0.00188	0.89	0.00322	1.52	0.21180	4,226.0	8,919.2	0.20	2.41	1.25	$\frac{1}{10}$ A. T.	
33	0.23392	0.00046	0.19	0.00018	0.08	0.23456	4,691.2	0.20656	0.00188	0.45	0.00298	1.42	0.21048	4,209.6	8,900.8	0.27	1.87	1.02	$\frac{1}{10}$ borax-glass.	
34	0.23324	0.00044	0.19	0.00082	0.35	0.23450	4,690.0	0.20422	0.00128	0.61	0.00536	2.54	0.21086	4,217.2	8,907.2	0.54	3.15	1.77	$\frac{1}{10}$ A. T.	
35	0.22962	0.00042	0.18	0.00068	0.29	0.23072	4,614.4	0.20278	0.00128	0.62	0.00422	2.03	0.20828	4,165.6	8,780.0	0.47	2.65	1.50	6 PbO.	
36	0.23416	0.00016	0.07	0.00050	0.21	0.23482	4,696.4	0.20444	0.00240	1.14	0.00346	1.64	0.21030	4,206.0	8,902.4	0.28	2.78	1.46	$\frac{1}{10}$ borax-glass.	
37	0.23428	0.00032	0.14	0.00040	0.17	0.23500	4,700.0	0.20432	0.00260	1.24	0.00290	1.38	0.20982	4,196.4	8,896.4	0.31	2.62	1.40	$\frac{1}{10}$ borax-glass.	
38	0.23382	0.00042	0.18	0.00062	0.27	0.23486	4,697.2	0.20480	0.00210	0.99	0.00332	1.58	0.21022	4,204.4	8,901.6	0.45	2.57	1.45	60 Pb.	
39	0.23452	0.00036	0.15	0.00036	0.15	0.23524	4,704.8	0.20488	0.00206	0.98	0.00284	1.35	0.20928	4,185.6	8,890.4	0.30	2.33	1.26	$\frac{1}{10}$ Borax-glass.	

TABLE VI.—Data of Assays of Slags and Cupels Obtained in Treating Zinc-Box Residues of Table V.  
(Duplicates of Table II.) Mr. Kehew.

First Scorification.				Second Scorification.				Charge for Fusing Slag.						Charge for Fusing Cupels.										
No. of Charge.	Weight Taken.	Weight of Pb. and PbO.	Borax-Glass.	Time of Fusion.	Weight of Button.	Lead Added.	Oils.	Time in Min.	Time Cupelling.	Weight of Slag.	Soda.	Glass.	PbO.	Argols.	Time of Fusion.	Weight of Button.	Soda.	Borax-Glass.	PbO.	Argols.	Glass.	Time of Fusion.	Weight of Pb. Button.	
																								Grammes.
27	46 A. T.	65	10	43	50	None.	2	35	22	27	55	15	10	30	2	40	17	44	15	60	2	None.	30	23
28	46 A. T.	65	10	63	46	None.	2	35	20	27	67	15	10	30	2	40	14	42	15	60	2	None.	30	19
29	46 A. T.	65	10	63	40	None.	2	35	11	15	66	15	10	30	2	40	21	21	15	60	2	None.	30	20
30	46 A. T.	65	10	41	41	None.	2	47	8	12	62	15	10	30	2	30	23	38	15	60	2	15	30	15
31	46 A. T.	65	10	71	51	None.	2	47	9	13	62	15	10	30	2	30	27	43	15	60	2	15	30	18
32	46 A. T.	65	10	30	23	None.	2	47	9	13	60	15	10	30	2	30	21	60	15	60	2	15	30	14
33	46 A. T.	40 Pb.	10	30	21	None.	2	34	11	15	48	15	10	30	2	30	19	46	15	60	2	15	30	17
34	46 A. T.	40 Pb.	10	31	40	None.	2	34	11	15	48	15	10	30	2	30	28	35	15	60	2	15	30	24
35	46 A. T.	6 PbO.	10	31	40	None.	2	34	8	11	57	15	10	30	2	30	29	28	15	60	2	15	30	21
36	46 A. T.	65 Pb.	10	72	12	None.	2	47	9	15	65	15	10	30	2	30	27	66	15	60	2	15	30	14
37	46 A. T.	63 Pb.	10	72	9	None.	2	47	9	15	60	15	10	30	2	30	20	98	15	60	2	15	30	17
38	46 A. T.	10 PbO.	10	72	14	None.	2	47	9	15	69	15	10	30	2	30	14	46	15	60	2	15	30	15
39	46 A. T.	60 Pb.	10	72	13	None.	2	47	9	15	72	15	10	30	2	30	25	42	15	60	2	15	30	16

perature in the back of the muffle was  $780^{\circ}\text{C}$ ., while in the front it was  $720^{\circ}\text{C}$ .

Charges Nos. 34 and 35 consisted of 0.05 Assay Ton of residues, mixed with 6 grammes of litharge in a 3-in. scorifier, on which were placed 40 grammes of test-lead, followed by a cover of 10 grammes of borax-glass. The charges were placed in a very hot muffle and the door was closed for 10 minutes, after which it was opened and the temperature allowed to fall to  $780^{\circ}\text{C}$ . As soon as the buttons had become covered, the slag was poured from them, but the buttons finally obtained were too large for cupellation and had to be rescorified. All slags and cupels were assayed as usual. Although charge No. 34 was fairly satisfactory, for some unaccountable reason the results for charge No. 35 were too low.

Charges Nos. 36 and 37 consisted of 0.05 Assay Ton of residues, mixed with 65 grammes of test-lead with a cover of 10 grammes of borax-glass, and charges Nos. 38 and 39 consisted of 0.05 Assay Ton of residues, with 10 grammes of litharge and 30 grammes of test-lead, having an additional quantity of 30 grammes of test-lead placed on top with a final cover of 10 grammes of borax-glass. Charges Nos. 36 and 38 were placed at the back of the muffle and Nos. 37 and 39 at the front. The temperature of the muffle was maintained as nearly as possible at  $790^{\circ}\text{C}$ . Owing to the door of the muffle having been opened too soon, charges Nos. 37 and 39 spit, the former quite badly. The slag from No. 38, which was at the back of the muffle, contained more gold than that from No. 39, which was in front, but the slag from charge No. 37, in the front, carried more gold than that of No. 36, which was at the back.

Considerable difficulty was encountered in parting the buttons when the ratio of silver to gold was 2.5 to 1, even after they were annealed and rolled thin. No difficulty resulted, however, when the ratio was 3.5 to 1. The data pertaining to these assays are given in Tables V. and VI.

Taking the best results, *i.e.*, charges Nos. 27, 28, 29, 30, 31, 32, 33, 36 and 37, an average of 4,699.2 oz. gold per ton is obtained; the difference between the highest and the lowest results being 18.4 oz., which corresponds to 0.39 per cent. Averaging the results for silver from these assays, the figure of 4,209.7 oz. per ton is obtained; the difference between the

highest and the lowest results being 29.6 oz. per ton, or 0.7 per cent.

In those charges in which litharge was used, Nos. 34, 35, 38 and 39 (omitting No. 39), the average determination of the gold was 4,697.3 oz. per ton, and of silver, 4,202.4 oz. per ton.

A comparison of the final average results obtained from the scorification-assays of the zinc-box residues, obtained by Mr. Hollis and Mr. Kehew, is given in Table VII.

TABLE VII.—*Comparison of Results of Scorification-Assays of Zinc-Box Residues, Obtained by Mr. Hollis and Mr. Kehew.*

	Gold.		Silver.	
	Ounces.	Value at \$20.67.	Ounces.	Value at \$0.50.
Mr. Hollis <sup>a</sup> .....	4,694	\$97,025	4,178.5	\$2,089
Mr. Kehew <sup>b</sup> .....	4,699.2	97,133	4,209.7	2,104
Difference.....	5.2	108	31.2	15
Difference percentage..	0.11	.....	0.75	.....

<sup>a</sup> Using coke fuel.

<sup>b</sup> Using gas fuel.

#### CONFIRMATORY WET ASSAY.

Mr. Kehew confirmed his results of the scorification-assay by the wet method of Mr. C. Whitehead, as follows:—0.05 Assay Ton of the residues was placed in a 250-c.c. casserole and 50 c.c. of water was poured over it, followed by 25 c.c. of strong nitric acid (sp. gr. 1.42). The casserole was then placed on a hot plate and allowed to stand for 2 hours. The residue after filtration should have consisted of gold and other insoluble material, but the button obtained by cupelling this residue contained silver, which necessitated a second cupellation, with the addition of C. P. silver. The slags and cupels were assayed, the final results being given in Table VIII.

TABLE VIII.—*Data of Assays of Zinc-Box Residues by the Whitehead Wet Method. Mr. Kehew.*

Number of Charge.	Gold.		Silver.	
	Total Content.	Ounces Per Ton.	Total Content.	Ounces Per Ton.
40	0.23468	4,693.6	0.20852	4,170.4
41	0.23496	4,699.2	0.20919	4,183.8

The results for gold as shown in Table VIII. are not as high as those of many of the scorification-assays; the results for silver are quite a little lower, and confirm the results obtained by Mr. Hollis.

From the foregoing experiments, the following conclusions may be drawn.

1. That the zinc-box residues must be in a sufficiently fine state of division to pass at least through a 200-mesh screen.

2. That assay balances, or balances of equal delicacy, must be used for weighing the residues.

3. That the results obtained by the scorification-assay, when properly made, are as accurate for the determination of the gold as those of any other method tried, and more accurate for silver than the Whitehead wet method or the combination wet and dry method.

4. That the most satisfactory charge is 0.05 Assay Ton of residues, mixed with 35 grammes of test-lead in a 3- or 4-in. scorifier, with 30 grammes of test-lead placed on top and a final cover of 10 grammes of borax-glass.

5. That a large quantity of borax-glass is absolutely necessary.

6. That the spitting of the charge can be avoided by placing the scorifiers in a very hot muffle, keeping the muffle door closed for at least 5 minutes after the charge had become fused, and then opening the door and reducing the temperature to from 780° to 800° C.

7. That the addition of a small quantity of litharge to a charge seems to lessen the danger of spitting, but when so added the results for silver will probably be low.

## Some Practical Suggestions Concerning the Genesis of Ore-Deposits.

BY MAX BOEHMER, DENVER, COLORADO.

(New York Meeting, October, 1903.)

DURING the last few years the *Transactions* of the Institute have been especially rich in discussions pertaining to the genesis of ore-deposits. Starting with the masterly treatise of Professor Pösepný, which successfully demolished Sandberger's theory of "lateral secretion from the rocks in the immediate vicinity of the veins," we have since had very thorough and able papers from the hands of Le Conte, Vogt, Kemp, Emons, Lindgren, Van Hise, Weed, Spurr, Rickard and others. It would seem that the combined efforts of so many able men should have solved the problem satisfactorily by this time. Indeed, Professor Le Conte, quite a while ago, expressed his definite opinion that the problem had been solved, so far as the geologist was concerned. However this may be, it seems certain that those whose particular business it is to examine and estimate the value of mines, and whose special work takes them underground in all the mining districts of the United States and other countries, are not wholly satisfied, but are still in doubt on many points relating to the origin of the ores in the various mines they are called upon to examine.

The subject is of the highest economic importance. A comprehensive theory, once fully established and understood, would be of immense and immediate practical value to all engaged in the mining industry. It would be of great value in formulating plans for the proper development of a mine and throw light upon the dark and undeveloped portions.

### THE ERUPTIVE ROCKS.

The first and most lasting impression made upon the mind of him who examines a mine or a mining district consists in the observation that in each and every case there is an intimate association of the ores with the eruptive rocks in the district. The mines in the Mississippi valley seem to be the only excep-

tion to this rule. It may be possible that further development or closer examination will still discover these, as was the case in Butte, Montana. In that district the porphyry dikes had not been especially noted, and were not generally known to exist until about ten years ago, when the frequent lawsuits necessitated a closer inspection of the ground. This intimate association of the eruptive rocks with the ores, true the world over, convinces the practical mind that there must be a genetic relation between the two. Upon a further examination of the district, it is frequently found that the ore-bearing veins are confined to certain areas, although the eruptive rocks continue for miles beyond. This, again, proves that other factors, not directly connected with the eruptive rocks, have been at work, and that such factors are local in their origin and effect because they are limited to certain portions of any given district. This fact, it would seem, precludes any attempt to explain the origin of ores which does not specifically account for the localization of the deposit.

The most obvious conclusion to be drawn from this intimate association of metalliferous veins and eruptive rocks is, that the source which furnished the material for the rocks is likewise the source of the waters which deposited the ores in the fissures. I take it as conclusively established that by far the larger number of ore-deposits are the work of underground water. Whether it is possible and probable that the source indicated is the true one must be demonstrated in a discussion of the circulation of underground waters.

#### THE SEA OF UNDERGROUND WATER.

All authorities agree that, as the earth cools, cycle after cycle, an increasing amount of water is absorbed and added to the sea of underground water. The water has worked its way downward ever since the cooling crust of the earth permitted its accumulation at the surface. And this process will continue until the earth cools to the centre and finally absorbs all the waters of the globe. The lower boundary or bottom of this sea of water is limited and determined by the heat-conditions of the interior. How far this limit reaches, it may be impossible to determine; but it is certain that it reaches the region of metamorphism, because this process of change of the sedi-

mentary rocks cannot be carried on without the presence of water. Nor is this water a local occurrence brought about by the opening of some deep-reaching fissures or by the subsidence of blocks of sedimentary rocks including masses of water. This superheated belt of water, near the bottom of the underground sea, must be universal in its scope, because the underlying granite is universal. The granite represents the ultimate stage of metamorphism, and cannot crystallize except out of magma in hydro-thermal fusion.

If this is correct, we have passed the zone of rock-flowage, which, according to Professor Van Hise, is the downward limit of underground water. Perhaps the bottom of the zone of rock-flowage is identical with the belt of hydro-thermal fusion.

It is a well-known fact, which has been repeatedly demonstrated by experiment, that rocks which require a temperature of 3000° F. for ordinary, dry fusion, can be made to assume a pasty or liquid form at a temperature of 500°, in the presence of sufficient water under pressure.

These conditions must exist at some depth at all points, if the existence of an underground sea of water, ever working its way downward, is admitted as universal. The extent in depth of this belt of hydro-thermal fusion it is probably impossible to estimate. But its waters, if not in active circulation, are ever moving and hard at work, leaching, extracting and keeping in solution the most soluble elements present. This region must be the seat of continual activity. The hydration and dehydration of minerals, the crystallizing and dissolving and re-crystallizing must be ever going on, until we have a steady, never-failing reservoir of superheated waters under great pressure and charged to saturation with the metals and the minerals present.

Nor can this reservoir be tapped from above and its burden of metals in solution be brought toward the surface without a catastrophe. The rending of a crevice or a fissure in the crust above sufficiently deep to reach this belt, and *sufficiently open to permit operation of gravitative stress upward*, alone, can start the circulation of these waters and bring them toward the surface.

It is correct, no doubt, theoretically, that the waters of a



river running into a lake at one end, and out at the other, affect and move the entire body of water, however deep. So it is true that the surface-waters, daily pouring down to join the underground sea, affect the entire sea to some extent.

But in the deeper region, where the rocks are tight and the water works its way slowly and gradually downward, friction most certainly has obliterated all effects, or nearly all, of hydrostatic head and stress by difference of temperature. The sea in its lower portions, therefore, must be regarded as stagnant. And a stagnant mass of water, although saturating the rocks throughout, can do but little leaching until the pressure and the heat reach a degree sufficiently great to dissolve and fuse the rocks. And this is the beginning of the belt of hydrothermal fusion, and here we find material sufficient to fill the veins.

The mining engineer in active practice will find it difficult to accept the theory of Prof. Van Hise, which makes the deposition of ores nothing more than an accidental and specially emphasized phase of the work of the underground circulation of meteoric waters. Such a theory would make a metalliferous vein out of all crevices and fissures at all points and at any depth.

It fully accounts for the existence of mineral springs and hot springs, with a possible and occasional admixture of the metals in their waters; but it fails to explain the localization of the deposits, depreciates the importance of the eruptive rocks, and does not convince us that the waters feeding the springs of the earth are the same which have deposited the millions of tons of ore in the great mining districts.

That the waters in the vadose region above the groundwater-level should be able to collect any respectable quantity of metal seems improbable, in spite of their greater activity in the more open rocks near the surface. These waters, reinforced by every rain and melting snow, do not search the rocks nor leach much value from them, because they naturally choose the more open spaces, enlarge them, and create their own channels, in which they remain permanently. When these waters reach the groundwater-level, they naturally spread out upon its surface, run off laterally (there being less impediment to their course above this level), and reach daylight again at a lower elevation.

That an underground sea of water, continuous and saturating the rocks throughout, really does exist, seems a corollary of the fact that the water, once held entirely at the earth's surface and above it, has gradually worked its way into the cooling earth; the only argument against it consists in the fact that a number of shafts have been sunk in wet ground for many feet, and at greater depth encountered a region free from water. Cannot this be explained by the probable existence of an impervious stratum at the point where the water ceased? I take it that such occurrence simply illustrates the very irregular contour of the groundwater surface. These shafts, if sunk still deeper, must encounter water again, if the underground sea of water does exist.

If, then, the existence of an extensive belt of hydro-thermal fusion be admitted, and if it is true that this belt is constantly supplied with water by a permanent underground sea, it will not be difficult to explain the mineralization of our veins and their localization. An extensive movement in the crust creates a profound and complicated set of fissures at certain points. These fissures reach the belt of fused rock. The blocks of solid rock above sink into the fused mass, and squeeze it into the fissures toward the surface. This makes the dikes and sheets of eruptive rock as we find them near the veins. Simultaneously or later, other fissures, not sufficiently open or not in proper position to admit the fused mass, start up the circulation of the waters and begin the making of a vein.

I have attempted to show that the waters in the zone of open fracture in the upper region do not search or leach the masses of rock, but that they flow through them in permanent and open channels; that the waters in the middle and lower zones of the underground sea are stagnant and not sufficiently effective; and that we must look still deeper for the source that could supply the immense masses of ore found in the veins.

The invariable existence and evident genetic relation of the eruptive rocks with our ore-deposits suggest the source. Is it the true one? I refer the matter to geological specialists for further consideration.

## Observations on Mother Lode Gold-Deposits, California.

BY WILLIAM A. PRICHARD, KALGOORLIE, WESTERN AUSTRALIA.

(New York Meeting, October, 1903.)

### INTRODUCTION.

(a) The mineral belt known as the "Mother Lode"\* may be described as a series of connected ore-bearing fissures lying within a zone or belt of country from a few hundred feet to a mile or two in width, and a hundred miles or more in length, traversing the counties of Amador, Calaveras and Tuolumne, in California.

(b) The valuable portions of the lodes comprising this belt are not confined to any particular fissure for more than a few hundred feet. The ore-bodies branch at frequent intervals from one fissure to another, parallel to it, maintaining, however, the general direction of the belt. The connecting branches and spurs are often more valuable than the lodes themselves.

(c) The ore-bearing fissures occur in all the rocks within the zone; but their structure is modified by cleavage and other physical peculiarities of the rocks in which the fissures occur.

(d) The distribution of the gold in the lodes is dependent, (1) upon the precipitating properties of certain minerals; and, (2) upon certain physical and structural peculiarities both in the fissure and in the walls of the fissure.

### GEOLOGY.

After a study of the excellent geological reports and maps prepared by the United States Geological Survey,† and after practical experience in working the ore-deposits in this field,

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\* Prof. J. F. Kemp (*Ore-Deposits of the United States and Canada*, pp. 361-375) gives a very full bibliography of the literature of Californian ore-deposits. It is impossible for the writer at the present time and place to make complete references, under the various headings, to all that has been written on these ore-deposits.

† "Mother Lode District Folio, California, No. 63."

the writer has come to the conclusion that the local rock-complex records three important epochs of structural disturbances directly connected with the gold-deposits:

1. A period of uplifting and intrusion of the rocks now constituting the mountains of the Sierra Nevada.

2. Later dioritic intrusions of smaller magnitude, resulting directly in the formation of fissures and ore-deposits.

3. A period of cooling and contraction, resulting in the faulting and crushing of the ore-deposits.

#### *First Period.*

The upheaval of the Sierra Nevada resulted in great folding of the overlying rocks. Sedimentaries and eruptives alike were thrust westward and bent into nearly perpendicular folds. This folding was followed by great faulting, which confused the entire rock series. The resultant horizontal geological section, after erosion, is a complex mixture of rocks of different ages. These rocks now appear on the map in interlocked strips, more or less lenticular in outline. Only individual portions of the mixture are conformable. The general lenticular outline has been cited as both the result and proof of the intrusive origin of some portions of the rock series. This is true, however, only in the case of a few eruptives of a period more recent than the epoch to which reference is here made. Owing to the great pressure, and the heat accompanying the intrusion of so large a mass, the overlying and adjacent rocks, which now form the bedrock complex, were in a molten or plastic condition when the folding and faulting occurred. The existence of open fissures through which solutions could pass, and the deposition of quartz along the fault-planes, were therefore impossible, except near the surface, which has been eroded since. These rocks were simply metamorphosed by great heat, pressure, and violent disturbance, received a schistose structure, and were impregnated with silica.

#### *Second Period.*

The next important event of the geological record was the intrusion of the diorites, now altered to "metadiorites,"\* which

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\* "Metadiorites" are altered diorites, largely quartz-diorites. *U. S. Geol. Survey*, Mother Lode Folio.

underlie a large part, if not the whole, of the Mother Lode belt. These rocks now constitute a broken band of irregular width along the east side of the ore-bearing lodes. The intrusions of diorite were small as compared to those of the first period, and may, perhaps, be considered as the after-effect of cooling and contraction of the superficial rocks overlying the still highly-heated interior. They appear to have emanated from the base of the solidified portion of the former intrusion. The later intrusion followed the general line of weakness previously developed. It is not a new idea to connect these eruptives with the ore-deposits of this district,\* and a similar relation to eruptives has been noted in many mining-fields of the world.†

*Circulating Solutions.*—The intrusion of these “metadiorite” dikes, though strong enough to rend the overlying rocks into a great network of fissures for hundreds of miles, was not accompanied by heat and pressure sufficient to obliterate the fissures thus formed. These fissures formed an outlet for the highly siliceous solutions ejected by pressure from the deep regions of great heat. The water of these solutions had found its way by percolation into the overlying rocks previous to the intrusion. This water, unable to escape suddenly from the intrusion, was imprisoned in rocks which subsequently became highly heated or possibly molten. The water worked in a general upward direction, this being the only direction of escape from the lateral pressure exerted by the molten or highly-heated rocks. In its passage, it dissolved silica and

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\* “Relation between Ore-Deposits and their Inclosing Walls.” H. W. Fairbanks (*Eng. and Min. Jour.*, March 4, 1893) stated that portions of California lodes where no dikes occur are poorly mineralized, and believes that eruptives only cause passages for deep waters. Lindgren (17th *Ann. Rep. U. S. Geol. Survey*, ii., 163) calls attention to the striking grouping of the Banner Hill and Grass Valley district along a semicircular contact of granodiorite with the older rocks. He also states that the veins of the Grass Valley district show some relation to the granodiorite which occurs on both sides along the whole extent of the district.

† *A Treatise on Ore-Deposits*, Phillips and Louis, pp. 213-214: “Copper- and tin-ores of Cornwall, in the Parish of St. Agnes, and in the western portion of Peranzabulor, were most frequently met with in the vicinity of ‘elvans,’ or quartz-porphyrates.” *The Mineral Resources of New South Wales*, by Edw. F. Pittman, pp. 56-57: “It seems possible that this enrichment of the ore-bodies is due to the intrusion of the igneous dikes, for all of the payable deposits hitherto worked have been in the immediate neighborhood of these intrusions.”

other minerals. These mineral-bearing solutions worked their way to the top of the fluid or semi-fluid rocks, and escaped intermittently through the plastic and solid rocks as these were repeatedly fractured by the progress of the intrusion.

*Mineral Precipitation by Vaporization.*—These ascending solutions, though highly heated, were kept from vaporizing by the great pressure to which they were subjected. In their upward passage through fissures, the solutions passed through rocks less heated and more rigid, and the fissures were more open and the exit for the solutions more free. The partial relief of pressure thus afforded permitted part or all of the water, still highly heated, to vaporize before reaching the surface. The vapor, of course, retained little or none of its former mineral contents, thus causing a concentration of the solutions left behind, which soon became saturated with silica and eventually with other minerals contained in smaller quantities. The excess of mineral was deposited on the walls until the fissures were closed by the precipitate thus formed and the flow of the solutions interrupted. At the ever-changing level in the fissure where vaporization took place, the solutions were in a state of great agitation, unfavorable to the formation of large crystals. The silica, being largely in excess of all other minerals, was first to precipitate, and was deposited on the walls of the fissure as massive quartz. The deposition of this silica tended to close the fissures. After this closing, conditions often became favorable for the deposition of crystals of quartz and the minerals contained in small amounts in the solution. This accounts for the occurrence, so common in large quartz-veins, of "vugs" or crystal cavities. These vugs are but the remnants of large cavities. Where the openings were small—generally where the walls were too weak to support large openings—the deposits were small. In the absence of sufficient quartz to sustain the cavities, lateral pressure and subsequent movements of the walls brought together the sides and left only a seam. Seams formed in this manner often contain minerals which had been deposited in them previous to the crushing of the cavity.

*Order of Precipitation in Cavities.*—The order of precipitation, due to concentration of solutions, depended, of course, upon the amount of each mineral present in the supply of solution

sent up from below, and also upon the saturation-point of that particular mineral. Many rare metals have long been known to exist in small quantities in all rocks, particularly in those of sedimentary origin; and silica forms a large part of most rocks. From the evidence afforded by the ore-deposits of the district, it appears that the solutions contained the various minerals in quantities much in proportion to their abundance in the most common rocks. These solutions apparently contained a large amount of silica, a small amount of pyrite, and traces of gold, silver, lead, zinc, etc. The solutions were no doubt saturated with silica before the escape of any steam; but they were far from being saturated with the precious metals. Evidence goes to show that the quartz was almost wholly precipitated before the precipitation of gold began. It appears that the small amount of gold present remained in the solution until the precipitation of silica and pyrite was nearly complete.

The writer has seen a number of crystal cavities in the stopes of the Keystone mine, Amador county, California. These "vugs" were found within large bodies of white "buck" quartz very poor in gold. When broken into, the cavities were filled with a solution containing a large proportion of semi-fluid hydrous mica (margarodite). Unfortunately, the solution in every case escaped, and a sample for analysis could not be obtained. From a single one of these cavities the writer obtained some beautiful specimens of quartz, pyrite, zinc-blende, galena, gold and silver alloy, and calcite, all crystallized. The general order of crystallization in this case was quite apparent. The large quartz-crystals were pretty well complete before the other minerals started to precipitate; but the quartz-crystals had continued to grow, enveloping some of the other minerals. A particle of gold weighing several grains was entirely enclosed in a large transparent quartz-crystal. After the quartz, iron pyrites was the next mineral to grow; but it had not developed completely before galena and gold crystals started to grow, and then zinc-blende, calcite and margarodite respectively. Gold and galena crystals are embedded in the pyrite, but they appear to have begun to form toward the end of the precipitation of pyrite. All of the minerals appear to have shown a greater preference for their own kind. Gold, pyrite, galena and zinc-blende preferred forming on each other

rather than on quartz; but calcite showed a distinct preference for quartz.

*Economic Importance of Pyrite.*—Pyrite does not appear to have been the precipitating agent for gold. It does appear, however, to have facilitated the precipitation of gold, by action as a depository for the precipitate. As described above, the gold, in the vug, chose to precipitate on pyrite in preference to quartz. This appears to be characteristic. The same causes that precipitated the gold appear to have precipitated the pyrite and various other minerals. Pyrite appears to have acted as a sort of connecting link between the gold in solution and the carbon in the slate. Pyrite, being largely in excess of gold in the solutions, started to precipitate before the gold. The gold appears to have been unable to keep pace with the precipitation of silica, and was therefore unable to keep in contact with the walls. The pyrite, being in greater abundance, formed rapidly when started, and no doubt projected through the film of silica on the walls for a considerable time after the walls were completely coated. The gold was thereafter deposited on the faces of the pyrite crystals, and included mechanically within them.\*

Vug-deposits are of but little economic importance, owing to the large amount of barren quartz necessarily deposited before the solution was sufficiently enriched for precipitation of gold by concentration. It rarely occurred that the solution, at the time of isolation, was sufficiently concentrated for the deposition of gold. For the deposition of ore-bodies of economic value, it was important that the excess of silica should not be too great, and that a direct contact of the gold-bearing solution should take place with a precipitating agent.

Where silica occurred in large excess, the excess was eliminated by precipitation in open cavities and in the cavities of crushed wall-rock. These openings varied in size from microscopic veinlets to great chambers. An excess of silica in a solution interfered with gold precipitation by coating the walls of the fissure with quartz. Contact between gold, in the solution, and the walls was thus prevented. The structural peculiarities of the ore-bodies in the district clearly point to the conclu-

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\* T. K. Rose, *Metallurgy of Gold*, p. 152.



sion that such a contact was ordinarily necessary. There are but rare instances where it is evident that gold was precipitated wholly by the concentration of the gold solution to the point of saturation. It usually occurred, in the ore-making process, that, after a large amount of silica had been eliminated by precipitation in cavities, subsequent movements crushed and tore apart the walls and their previously deposited inclusions. Fresh cavities were thus formed, into which the remaining gold solutions were squeezed and brought into contact with rocks possessing precipitating properties. Such movements account for the crushed, banded, and jointed structures which often characterize these lodes. When fissures were disturbed in this manner, the solutions were evaporated, and their gold contents were deposited on the walls and in the new cavities before it was possible for fresh silica-bearing solutions to come up from below and again cover the walls with quartz. The gold required but little space, hence very small openings were capable of receiving the gold from a large amount of solution. It was different, however, with the silica. The small openings in the walls offered comparatively little space for silica, and but little deposition was required to close them. The larger openings, made at the same time, received the bulk of the silica. Thus the narrower openings, if situated so that much gold solution passed through them, received with each filling a larger percentage of gold per unit of space and a smaller percentage of silica than the larger openings. When this process was repeated by successive openings, rich ore-bodies were formed where the favorably situated small openings occurred; and where the openings were larger, large quartz bodies were built up, in which the seams of gold were frequently too widely separated by quartz to give economic importance to the deposit.

It may be concluded from this that the portion of the fissure that opened by small increments was more favorably situated for enrichment than that part opened by large increments.

Rich ore-bodies of great width are found whose seamy structure shows that they were built up by small periodical increments next to one or both of the walls. The parts of fissure-structure most likely to cause small increments of opening are those parts which are not in the plane of sliding or tangential

movement of the walls. In this district the cleavage-planes of slate and schists are nearly vertical, and the fissures generally dip east. Many of the fissures in these formations have kept their general direction by a zigzag course, alternate portions following the more or less vertical cleavage-planes. In the portion of a fissure where an upward movement of the hanging-wall took place, a sliding contact resulted along the smooth cleavage-planes; but in the part of the fissure which lies across the cleavage-planes the walls have been simply pulled apart. The writer has observed in a number of mines that the richest ore-bodies in the Mother Lode mines lie across the cleavage, in these pulled-apart portions of the fissure.

*Precipitation of Gold by Carbon-Bearing Slates.*—It is evident that a lesser degree of concentration of solution was required for the precipitation of gold when solutions were in direct contact with certain wall-rocks. The facts strongly favor the conclusion of a sedimentary deposition of at least a large portion of the greenstones in the district, and there is no doubt about the sedimentary origin of the black clay-slates. Any precipitating properties of these rocks on gold may, therefore, in view of all that has been written on the subject, be attributed to organic matter in them.\* A great many of the greenstones are interbedded with clay-shales, a crosscut showing a banded structure. The thicker bands of greenstone are green and coarsely crystalline, but the thinner bands vary in texture and color in all degrees. These bands alternate in some places in great numbers.

Lodes, crossing rocks which have this banded structure, show striking variations in value, corresponding to the amount of organic matter in the rocks, as indicated by the color,—the blackest color denoting the most carbon. The writer has noticed a corresponding variation in the reducing power of these different wall-rocks on litharge in assaying. Sulphur-contents being uniform, an assayer may be guided by the color of the rock in reckoning the amount of reducing agent required in assaying samples of these rocks.

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\* T. A. Rickard, *Trans.*, xxii., pp. 314-315; xxvi., pp. 977-979, and *Eng. and Min. Jour.*, vol. lx., pp. 561-562. C. R. Van Hise, *Trans.*, xxx., pp. 94-95 and 118-120. J. R. Don, *Trans.*, xxvii., p. 569. E. E. Lungwitz, *Eng. and Min. Jour.*, vol. lxix., p. 500.

The total absence, or small amount, of organic matter in these bedded greenstones, and a frequent conglomerate or brecciated structure, argue that they were deposited with a rapidity which precluded the existence of organisms. The conclusion that most of these greenstones are of sedimentary origin is not contradictory to the opinion that the greenstones of this region are largely derived from tuffs and eruptive detritus. Some of them may also be derived from lava-flows on the sea bottom during sedimentation.

The most valuable ore-bodies in this district owe their enrichment to the presence of the black carbon-bearing slates. Yet even in these slates only small portions of the lodes are valuable, for the reason that after the fissure was opened the slate walls became coated with silica before the solution contained sufficient gold to make a precipitate of economic importance.

When a movement of the walls crushed the quartz-lining and permitted a solution containing an accumulation of gold to escape from the fissure into or along a slate wall, the gold and pyrite were immediately precipitated on the slate. When such a solution escaped into fractured greenstone schists, it diffused its gold throughout a greater mass of material before precipitation was effected. Precipitation, in this case, may have been caused by organic matter in the rock itself, or by infinitesimal organic particles which had been carried in mechanically from the black slate. It is most probable that some of the gold was precipitated by chemical or physical means, as when imprisoned in a vug, as described above. The gold, as well as the pyrite, decreases in amount as the distance from the main fissure increases.\*

This variation in the precipitating properties of the different types of wall-rocks accounts for the occurrence of ore in pockets, and in comparatively small rich deposits in the black slates, and for the wide dissemination of values in the greenstones. Nature has provided the black slate as a concentrator.

### *Third Period.*

Following upon the dioritic intrusions and the formation of the existing lodes came the cooling and contracting period,

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\* Lindgren, *17th Ann. Rep. U. S. Geol. Survey*, ii., pp. 124-125 and 146.

unaccompanied by heated solutions and mineral deposition. The more rigid rocks contracted almost without rupture. Their contraction was adjusted by movements along previously formed fissure-planes, and by distortion and settling of the soft, weak, clay-slates. These movements crushed and faulted many of the lodes, and permitted the soft Mariposa slates to settle down vertically in folds between the walls formed by the more rigid slates and greenstones. This settling of the slates detached and carried downward large portions of paying lodes, and also formed many recent fissures that are marked only by great sheets of black mud or crushed slate (gouge), mixed with fragments of quartz-lodes. These recent fissures are very misleading, when one is deciphering the vein-structure. They are, however, often of considerable importance on account of their mineral contents, consisting of large segments of quartz-veins, quartz-boulders, and finely crushed or "sugar" quartz.

A good understanding of these recent fissures is important, in order to save much costly development on pseudo-lodes. Extravagant litigation might have been avoided in the district in more than one case by an understanding of the difference between lodes and fault-planes. Lawsuits have been instituted by parties owning the outcrop of a fissure that has been formed ages since the deposition of a certain ore-body, to which it could be traced by a trail of crushed slate. A similar connection could be traced, if desired, to almost any other ore-body in the vicinity along one of these innumerable fissures.

*Structural Peculiarities.*—There are certain structural peculiarities along the Mother Lode that have an important bearing upon the distribution of gold in the lodes. The celebrated old mines of this region are those in which a rich ore-body outcropped at the surface. Any miner could follow one of these ore-bodies in depth, so long as the pay-shoot was continuous. There are other rich deposits, however, that are found only by accident or by a mastery of the structural geology. With these deposits ordinary prospecting-methods are inconclusive. Particularly is this true of the class of deposits where the principal enrichment is wholly within the walls, unattached to the main quartz-lode, which at that particular point may or may not carry values. Such detachments were caused by recent movements of the walls, resulting in the formation of great sheets of crushed material, called "casing" or "gouge."

Mother Lode mines are commonly associated with a broad belt of black, carbon-bearing clay-slate, termed, by the United States Geological Survey, the "Mariposa Formation." There are two reasons for this association:

1. The soft slates offered less resistance to fracture than any other rock in the region, hence more fissures reached the surface through them. Flat, dipping fissures in other rocks were deflected upward by the easily opened vertical laminations of the slates.

2. The organic contents as well as the physical structure of the slates generally favored precipitation of gold, hence a large part of the lodes occurring in this slate-formation are of economic value.

As stated at the beginning, the veins constituting the Mother Lode occur in all materials of the mineral zone, and it is important to be able to determine which portion of such a lode, or what lode (where there are many), is most likely to be sufficiently enriched for profitable extraction. With this object in view, a few of the types of ore-bodies are herein described, and the foregoing theories applied to these special cases.

*Regular Veins.*—Well-defined straight fissures are generally filled with a large, regular, clear quartz-body, or a brecciated mixture of quartz and wall-rock. Gold in clear quartz occurs only in occasional patches or seams. This peculiarity is accounted for by the regularity of the walls having caused all movements to affect the whole fissure alike, thus minimizing the possibilities for the isolation of solution, or for the fracturing of walls, and contact of solutions with naked walls. Every disturbance of the walls caused a free outlet for solutions from the greatest depths; thus great quantities were brought into the fissure. Deposits in these fissures contain more crystal cavities than any other deposits in the locality.

*Irregular Veins.*—The most valuable veins of the Mother Lode are those occupying fissures whose course is zigzag or winding. Every movement of the walls of these fissures necessarily caused an opening of certain portions of the fissure and a crushing of certain portions of the walls. This, as described previously, favored intimate contact between solution and country-rock, resulting in enrichment of certain portions of the vein and impoverishment of other portions. The rich, banded

or "ribbon"-rock of the district occurs in such fissures. Its structure is due to the opening of the fissure intermittently, and the building up of the ore-body by small increments.

In hard rock, a slight turn in the fissure was sufficient to cause these openings and crushings; but in flexible schists greater irregularities were necessary to produce the same results.

*Crushed Zones.*—Zones of crushed country-rock are the result of the movement of the walls of an irregular fissure, whose projecting irregularities were crushed into a brecciated mass and drawn into the fissure, or of the fissuring of a schistose or cleavable formation too yielding to fracture with a clean break.

To the latter class belongs the so-called "mineralized country-rock" or "gray ore." Thin sections of gray ore from the Keystone mine, prepared and examined by T. J. Hoover,\* showed a minute vein-structure, with crystallized gold and pyrite in the quartz. The structure in a thin section appears similar to the structure as it appears on a larger scale in a hand-specimen. The country-rock is fragmentary, and but very little altered by metasomatism, if at all.† The gray color may be due to the presence of extremely minute quartz-veins. These deposits are of great size in the Utica, Zeile, Mahoney, Keystone and many other important mines of the district. The gold in them is much more uniformly distributed than in the black slate deposits, and the value is much lower, except in rich leaders of quartz.

The deposition of gold in crushed zones may be accounted for by concentration of solution. Great widths of schists have been crushed and the space filled with quartz; but only comparatively small portions have received sufficient gold for profitable mining. The gold in these portions lies principally in leaders or stringers of quartz.

The first crushing permitted a flow of siliceous solution and the deposition of silica in the minutest cracks of the schist. As these minute veinlets were closed by the deposition of

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\* T. J. Hoover assisted the writer in tracing out the structural peculiarities of the Keystone and other Mother Lode mines.

† Lindgren, *17th Ann. Rep. U. S. Geol. Survey*, Part ii., pp. 124-125 and 146. Van Hise, *Trans.*, xxx., p. 159.

silica, the gold, not yet being sufficiently concentrated for deposition, remained in solution and retreated into the larger openings. With each successive crushing of the schists they became more siliceous and harder, and the crushing extended farther into the softer walls. With every crushing and filling the gold solution became more concentrated, until the gold was finally deposited with the silica. These deposits filled the cracks of the silicified crushed zone, thus forming rich leaders in a siliceous schist, a form of deposit found in many parts of the world. By the process just described, a great width of country-rock was crushed and silicified; but the deposition of gold in economic quantities was confined to comparatively small portions, throughout which, however, the small gold-bearing leaders are pretty uniformly distributed.

Some "mineralized country-rock" deposits may be due to the concentration of gold-bearing solutions containing but little or no silica. A concentration of such a solution would make a richer gold-deposit than one containing a large proportion of silica. The most siliceous solutions may have derived their silica from the more acid rocks. Those containing a lesser percentage of silica may have received their mineral contents from the more basic rocks.

The crushed greenstone-deposits in the Utica and Zeile mines doubtless owe their value to the small amount of silica which was present in the gold-bearing solutions, while the rich quartz-deposits of the Gwin, Kennedy, Argonaut, Central Eureka, South Spring Hill, Keystone, Bunker Hill and other mines owe their value to the segregating effect on highly siliceous solutions of certain local variations in the structure and composition of their wall-rocks.

## The Garnet-Formations of the Chillagoe Copper-Field, North Queensland, Australia.

BY GEORGE SMITH, SYDNEY, NEW SOUTH WALES.

(New York Meeting, October, 1903.)

CHILLAGOE is situated in latitude  $17^{\circ}$  S., about 138 miles from the port of Cairns, with which it is now connected by rail. A township has been established on the banks of Chillagoe creek, a perennial stream of irregular volume crossing the field almost at right-angles, nearly midway between the mining-camps of Calcifer and Mungana, which have been formed at the south-eastern and western extremes of the field respectively. This situation was selected because of the water-supply and its proximity to the site most suitable for the erection of reduction-works. By its central position it commands the whole of the mines. The altitude above sea-level is about 1200 ft., whereas that of Calcifer and Mungana is 1400 ft.—a difference sufficient to account for a materially higher temperature at Chillagoe. The surrounding limestone bluffs add by radiation to the heat, and, at the same time, prevent the cool breezes from passing freely through the town; consequently, the temperature, which is high throughout the year, and especially so during the summer months, is here generally very oppressive also.

The general trend of the field is southeast and northwest, and its length about 18 miles, with an average width of 5 miles; the approximate area is thus 90 sq. miles. The surrounding mineral-district is, however, much more extensive, and numerous other mines, spread over wide areas and producing a variety of minerals, exist in the same or parallel belts of mineralized country, of which Chillagoe proper covers but a very small portion.

Immense bluffs of limestone, wollastonite and conglomerate, rising in places to nearly 300 ft., and sometimes nearly a quarter-mile across, give the field a very picturesque and interesting appearance. The limestone has been classified as Devo-



nian, and, with one exception, which is of a distinct type, and therefore beyond the scope of this paper, the whole of the mineral-formations occur at or near its junction with igneous rocks.

In the neighborhood of Calcifer, the prevailing rock is hornblendic granite, probably of later origin than the limestone, as indicated by the alteration of the latter to calcite at the contact. Towards the west and northwest, where Mungana and Redcap (the third mining-camp) respectively are situated, a distinct change takes place in the geological features. Reversing the order, the west and northwest points are represented principally by felsitic rocks, in contact with which the limestone retains much of its crystalline structure, and is often highly fossiliferous; but as the southeastern extremity is approached, the felsites are replaced by granites, fossilization disappears, the limestone loses its structure and gradually becomes more crystallized until, finally, it is represented by calcite only. The southeastern end appears, therefore, to have been a much more active igneous center, and it is in this locality that most of the garnet-formations occur.

The lode-formations of the field are numerous, and represent several very interesting varieties; but it is proposed herein to refer specially to the occurrence of the garnet type, which constitutes about half the total number. Many of the formations of this type are intimately associated with large masses of magnetite, which is regarded as of analogous origin, and is, therefore, considered with the garnet. Secondary ore-deposits enclosed in limestone, whose origin may be clearly traced to these formations, will also be included.

The magnetite occurs only in association with the garnet, the two merging imperceptibly into each other, but there are many garnet-formations which show no magnetite; the latter is, therefore, not an essential accessory. Both occur in crystalline masses, generally with large, bold outcrops, and sometimes showing fine crystallizations. Single crystals of magnetite up to 10 in. in diameter have been met with, and equally large crystals of garnet, the common form in each case being the rhombic dodecahedron.

Very little work was done on the magnetite, though it evidently possesses the same ore-bearing characteristics as the

garnet. As a rule, where magnetite was met with in underground explorations, it was invariably found in association or closely connected with the copper-minerals, especially chalcopyrite, and, in consequence, was looked upon as a good indication of increasing copper-value. The garnet is much more prevalent and extensive than the magnetite; it has been proved to be as much as 200 ft. in thickness, and in some instances may be traced on the surface at intervals for considerable distances, forming lenticular masses of varying dimensions, sometimes disappearing beneath the surface over long spaces, and outcropping again and again, often in fairly direct line. Whether breaks in continuity occur is not known: early workings on the ore-deposits in the limestone previously mentioned have exposed garnet-rock at shallow depth, which is evidently a continuation of known occurrences beyond where the outcrops ceased. Apparently, therefore, the rock is sometimes more persistent in underground extension than is apparent on the surface.

The garnet-formations contain both copper- and lead-minerals, which are practically distinct from each other and confined to separate occurrences, though the gangue in each case is identical. None of these lead-formations, however, were explored by the writer, who will therefore deal only with the formations worked for copper.

The whole of the formations occur in close proximity to the limestone, generally, though not necessarily, along its contact-lines with the granite, the latter forming the foot-wall. There are instances in which they are enclosed on both sides by granite or felsite, aside from those just referred to as found at shallow depth below the surface. The latter are to all appearance surrounded entirely by limestone, which they seem to have penetrated as dikes. It is true, the workings upon them are superficial and insignificant, and the fact that igneous rocks have not been exposed in contact is not conclusive evidence that they do not exist at these points; their apparent absence, however, has raised doubts as to whether their presence was essential to the origin of the garnet.

The upper portions of the formations show evidence of much weathering, and the paucity of payable ore above the permanent water-level in many cases is due to its partial removal by

leaching. The water-level varies considerably with the season; during the early part of 1899, when some 55 in. of rain fell in four months, the ground-water of the lower-lying parts of the field came to within 9 ft. of the surface. The surrounding granite has become, through decomposition of its feldspar, so porous as to permit of the free underground passage of surface-waters, which thus find easy entrance into the formations. The lode-water may thus represent the drainage-area of the immediately surrounding neighborhood; but good natural drainage and drier weather soon reduce the quantity sufficiently to make it controllable by moderate pumping-appliances. Though many of the formations which have been subjected to the repeated leaching have lost much of their original copper-contents, there are others whose surface-ores, through a natural concentration of richer salts, are of higher value than the unaltered portions of the shoots. This is particularly true of the outcrops, or higher parts of the formations, which are above the reach of ground-water. The leaching-operation is still in progress, and is now greatly assisted by mining-works, which in opening up the ground hasten oxidation and permit a freer circulation of meteoric waters.

Of the secondary ore-deposits which occur in the limestone, I shall describe only those known to be connected with the garnet. Other deposits, of greater extent and importance, coming within this classification, but having no recognized connection with the garnet, will require a passing reference only. These include ore-bodies which evidently occupy pre-existing spaces in the limestone, and therefore come within the category of cave-deposits. The material forming the gangue of these is composed, principally, of a ferruginous and manganiferous clay (the manganese being present only where the ore is richer); and there seems little doubt that it was carried down by surface-waters, the filling being hastened by the weathering of the limestone, which is everywhere seen in process of decay.

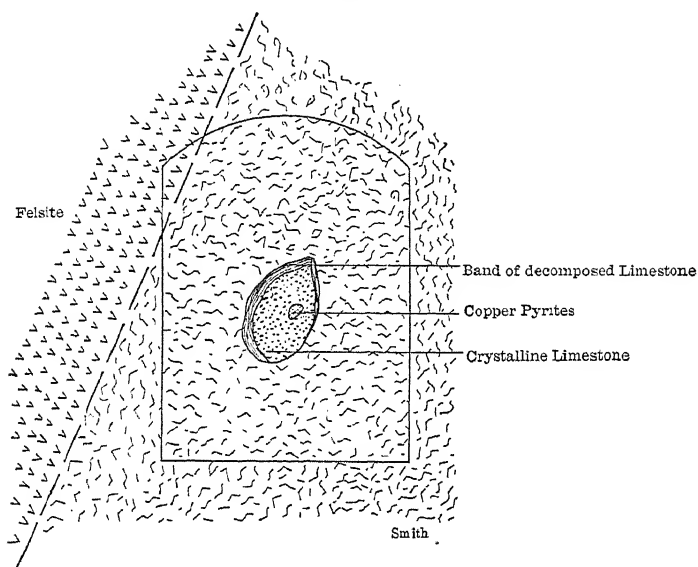
In the early history of the field it was customary to work as much as possible from the surface; and any superficial deposits of payable ore found were extracted, but few attempts being made to prove their downward extension. Several such deposits found in the limestone were exploited by this primitive

method of mining, which simply called for the extraction of the ore above water-level, or as much as could be easily and profitably won, before changing the scene of operations. Many small excavations thus made still remain partly open, and have been largely instrumental in conveying the impression that the ore occurred only as isolated surface-bunches, without downward extension. Investigation, however, would show that they are connected, in close proximity or in actual contact, with garnet-formations which, at these points, do not extend to the surface; although, owing to the wearing down of the limestone, they have been brought sufficiently near to become subjected to surface-influences. It is only at such places that these deposits have been met with; and their origin may be attributed to solutions which had entered the formations, possibly at higher levels, and dissolved their copper-contents in passing through to lower ground. Being charged with the acid products of decomposition of the sulphides, their action on the limestone would be considerable, and, assisted by continued denudation from above, they would in course of time gradually break through the overlying crust. A connection once made with the surface atmospheric influences would hasten the deposition, by replacement of the limestone, of the copper held in solution. Gypsum occurs very sparingly, but interesting examples of the replaced limestone may be seen, the same stone sometimes showing malachite and limestone on opposite sides, gradually merging into each other. Fig. 1, representing the replacement of the limestone by sulphides, is taken from a formation of different type for the purpose of illustration; no such occurrence was noted in connection with the garnet. In this case, a drift following the hanging-wall in solid ore, composed of pyrrhotite, chalcopyrite and blende, encountered at 150 ft. what appeared to be a boulder of limestone. As driving was continued, however, it was seen to be a spur of that rock which gradually widened, the ore diminishing correspondingly, until but a foot of thickness remained, when driving was suspended for the time being.

The origin of similar occurrences of garnet elsewhere has been ascribed to contact-metamorphism, by which it is assumed that molecular rearrangement or metasomatic replacement of the limestone has been induced by the action of superheated

steam or vapors evolved from the igneous rocks at the time of their extrusion, the copper being derived from the same source in the operation. The largest garnet-formation at Chillagoe explored by the writer (about 200 ft. thick) is enclosed on either side by granite; limestone (calcite) is seen on the surface within a short distance, but whether in actual contact with the garnet at any point was not determined. A dark-colored enclosure, intersected by a tunnel near the center of the formation, was microscopically examined by my friend, Mr. J. C. Moulden, A.R.S.M., an ardent and able petrologist, who classifies it as a

FIG. 1.



This ore assays: Cu, 6; and Zn, 12 per cent, with Ag, 6 oz. per ton.

FACE OF DRIFT, SHOWING BODY OF CHALCOPYRITE (WITH ZINCBLLENDE AND PYRRHOTITE, IN LIMESTONE).

“granulitic, compact rock, composed of biotite, plagioclase and quartz, resembling the segregations of more basic material which occur in many granites.” It seems difficult to understand how such a formation could have been produced either by replacement or rearrangement of limestone, especially as granite may be seen in close proximity in actual contact with the limestone, which shows no other change than that of crystallization. The effect of intrusion upon the limestone is well illustrated by one of the smaller bluffs, which is completely cut

through at right-angles by granite. The apex of the latter shows on either side of the bluff, some distance above the present surface, but less than half-way to the summit. The limestone is crystallized at the contact, but no garnet is visible in the immediate neighborhood. There would thus appear to be no intermediate stage of alteration between the crystallization of the limestone and its complete change to garnet, nor is there any evidence of replacement. In the writer's experience, where replacement in lodes or of wall-rocks has occurred, some indication of the alteration can generally be seen; indeed, its progress may sometimes be traced at various points, from the incipient stage, represented by mere discoloration of the material by impregnating solutions, to complete replacement. If contact-metamorphism allows for a similarly gradual replacement, one might expect to find some recognizable evidence of the change, such as partial alteration of the limestone to garnet; but, if this exists, it escaped the writer's notice.

Wollastonite, which is also mentioned as the result of contact-metamorphism of the limestone, is abundant at Chillagoe, but it was not known to contain any mineralization, while the garnet was never without it. If the same origin and method of metamorphism be common to both rocks, it is not apparent why they differ so widely in metallic contents.

The close association of the garnet with the limestone apparently points to an essential connection between them, yet the former rock, occurring elsewhere and presumably without such association, has been unhesitatingly classified by eminent authorities as of igneous origin.

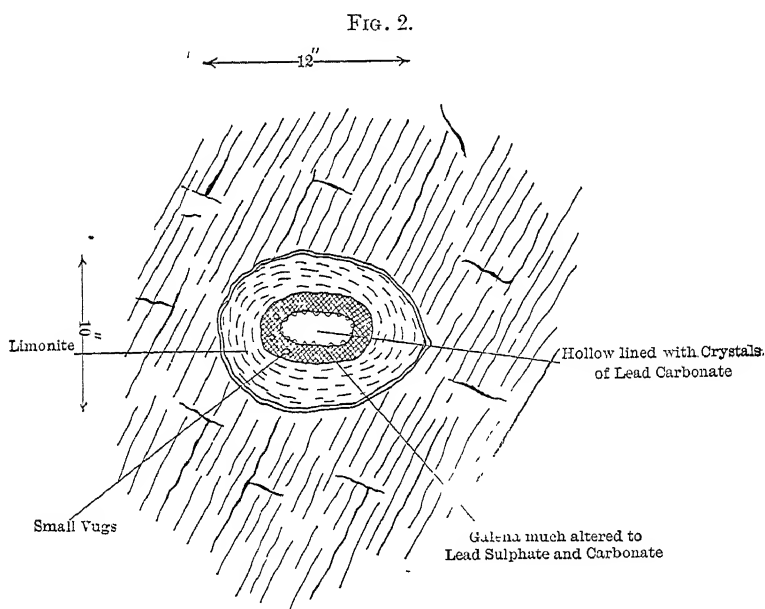
Samples of the limestone (calcite), representing very large quantities near the center of the field, were analyzed with the following results:

		CaCO <sub>3</sub> . Per Cent.	MgCO <sub>3</sub> . Per Cent.	SiO <sub>2</sub> . Per Cent.
No. 1,	. . . . .	99.0	0.2	0.65
No. 2,	. . . . .	99.2	nil	0.20
No. 3,	. . . . .	97.3	0.1	1.25

These samples were approximately representative of the bluffs they were taken from, and are sufficient to show that the garnet is not the result of the rearrangement of such limestone as the only cause.

The explorations in the garnet-formations, upon which these

notes are based, were for the most part superficial, having been checked for some time by heavy water, which was generally met with at about 50 ft., before pumping-appliances arrived on the ground. Deeper sinking was subsequently carried on, and several shafts were sunk to 150 ft., the level chosen for opening-out; but, unfortunately, an abrupt suspension of operations was insisted upon, which practically precluded any observations at that depth. The evidence available of the contact of the garnet-limestone was therefore confined almost entirely to shallower levels. The two rocks were found to be



CROSS-SECTION OF A SMALL PIPE OF LEAD-ORE IN LIMESTONE.

separated by clayey material ("dig") from about 1 to 10 ft. in thickness, and sometimes impregnated with copper-salts leached from the formations, even to the extent of being as rich in that metal as the adjacent portions of the formations. Coarse gravel, mostly angular, though sometimes water-worn, was freely distributed through the clay, having apparently been carried downwards and mechanically deposited by surface-waters, which had eroded spaces in the limestone in passing between the two rocks. Some of the clay also was, no doubt, carried downwards in like manner, but much of it was probably pro-

duced by movements of the hanging-wall (limestone). Very little movement appeared to have taken place on the foot-wall, the garnet being separated from the granite by a very thin seam of fluccan. In only one instance was the garnet-limestone contact seen at 100 ft.; little or no movement had taken place here, and but little clay was present. A chimney-like space had been formed at the contact, which continued in an unbroken line from the surface, and through which gravel, etc., had passed from the higher levels. The upper portion of this space was occupied by copper carbonate and silicate, which continued downwards for some distance, gradually "petering out" at 50 or 60 ft. This ore had evidently been leached from the outcrop and carried downwards by surface-waters.

The section of a small pipe of ore, shown in Fig. 2, is an interesting example of this shallow deposition in the limestone.

Microscopic examination of a thin section of the garnet, which in this case contained an unidentified mineral, was also kindly made by Mr. Moulden, who classifies it as "Eclogyte, consisting mainly of a rhombic pyroxene (bronzite) partly altered to bastite and garnet." This, according to the views of leading British petrologists, is, unquestionably, an igneous rock.

The following analysis of a normal variety of the garnet is the only complete one at the writer's disposal:

*Analysis of Garnet.*

	Per Cent.
SiO <sub>2</sub> , . . . . .	40.5
Fe <sub>2</sub> O <sub>3</sub> , . . . . .	11.1
Al <sub>2</sub> O <sub>3</sub> , . . . . .	16.7
CaO, . . . . .	31.6
CuO, . . . . .	0.3
MnO, }	
MgO, }	traces
	<hr/> 100.2

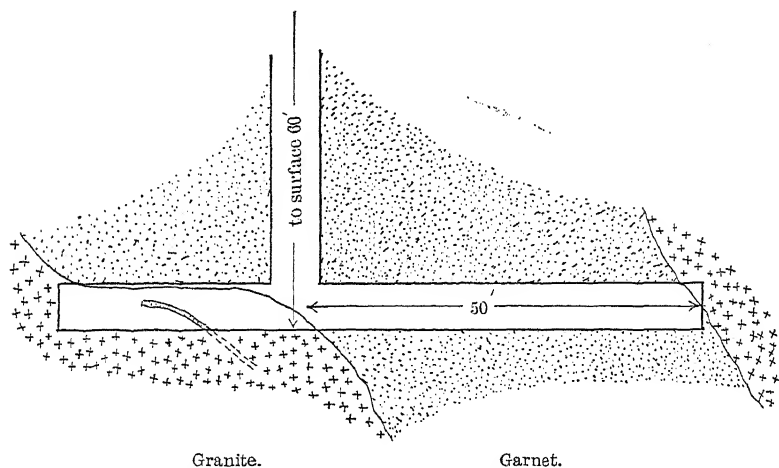
This analysis shows that, like other garnets, this one is a fusible compound; and, if regarded as an igneous rock, its position in relation to the enclosing rocks would not be difficult to account for, as its extrusion would naturally follow those lines of least resistance which, as we may reasonably assume, were generally found at the junction of the granite and limestone. The occurrence of the magnetite could then, per-



haps, be explained as the result of magmatic segregation of the excess of iron present, beyond that required for the garnet. The presence of the copper could possibly be accounted for in a similar manner.

A bulk-sample of Chillagoe magnetite, taken by the writer from a large outcrop, was analysed by Mr. W. C. A. Goyder (late Government Analyst of South Australia), who reported that, after the separation of 1.3 per cent metallics, containing 90.5 per cent copper (equivalent to 1.17 per cent metallic copper in the original sample), the following results were obtained:  $\text{Fe}_2\text{O}_3$ , 96.96;  $\text{Al}_2\text{O}_3$ , 0.48;  $\text{H}_2\text{O}$ , 0.15; Insol., 0.84 per cent, and a trace of copper; total, 98.43 per cent. The analyses showed also: silver, 7 dwt. per ton and a trace of gold.

FIG. 3.



CROSS-SECTION OF A GARNET FORMATION, ENCLOSED IN GRANITE.

Although all the iron is reported as ferric oxide, a small portion of it was present in the metallic form.<sup>1</sup>

The cross-section, shown in Fig. 3, represents one of the garnet-formations, which, at the point illustrated, is enclosed on both sides by granite.

The spur of garnet is evidently an offshoot from the main body, and bears a striking resemblance to a miniature dike, which in its upward course has followed a joint in the granite, becoming suppressed before reaching the surface.

<sup>1</sup> SECRETARY'S NOTE.—This paragraph does not appear in the pamphlet edition of Mr. Smith's paper, as the information was not received until after that edition had been printed.—R. W. R.

Near Calcifer, the remains of what appears to be an overflow of the garnet may be seen, enclosing a piece of calcite a foot or so in diameter. The position of this enclosure is difficult to account for, unless it be a boulder of altered limestone which was caught up by the garnet while still in a plastic condition.

The formations are very irregular in their dip, as are also the ore-bodies in every respect. As the work of exploration proceeds, the latter will probably be found to occur in semi-detached or isolated areas of "orey" ground, the intervening spaces being very poor; but it is believed that these areas, however irregular, possess a more or less systematic arrangement in their downward extension. The copper was found to be disseminated through the ore-bearing portions in small patches of solid ore, the largest being rarely above a foot in diameter. The average value of the "orey" ground varied from below 1 to above 10 per cent of copper, with a few ounces of silver and up to 6 dwts. of gold, per ton. Free gold was seen in the clear garnet in one formation only.

Speaking generally of the field, it may be added that the occurrence of the surface-deposits of ore previously described has been regarded by some as proof of the superficial character of the whole district, though it should be clear that such deposits, however uncertain and shallow, can have no bearing on the permanence of the formations they were derived from. The presence of the limestone has simply rendered the conditions suitable for the deposition of what, under less favorable circumstances, would have been carried away, as has happened in the case of many of the other formations in which the same conditions do not prevail. The effect of leaching on the formations has also promoted a feeling of distrust which the history of the field, quite apart from its actual development, unfortunately fully merits. But it is surprising that such areas of impoverishment, resulting from so simple a cause, should ever have been seriously accepted as indicative of what deeper explorations might be expected to produce.

So little information of what exists below the permanent water-level could be gathered, that it would, perhaps, be hazardous to express any definite opinion upon the probable results of deeper explorations. It may be said, however, that where the

ore-bodies were opened below the zone of oxidation, chalcopyrite was encountered. It is true the quantity was sometimes very limited, but it was generally commensurate with the exploratory work which led to its discovery. It would be unreasonable to expect the whole of the formations to develop satisfactorily. Some may, and probably will, prove very disappointing, as is so often the case with similar lodes in any mineral district. Those which the writer was permitted to develop partially, however, presented, up to the time of his leaving the district, no evidence indicating superficial deposition only; on the contrary, the general results obtained were such as to justify the belief that under capable and unfettered direction, centered on the field, and removed entirely from speculative requirements, the completion of systematic exploration in depth will ensure the realization of earlier and more optimistic anticipations of the value and future of the mines.

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### School Laboratory-Work: A Free-Milling Gold-Run.

BY ROBERT H. RICHARDS AND E. E. BUGBEE, MASS. INST. TECH., BOSTON, MASS.

(Albany Meeting, February, 1903.)

FRIENDS often ask how accurate the work of our little stamp-mill is, or express the opinion that a little mill cannot do good work. As a reply we would like to place on record the results of a few tests.

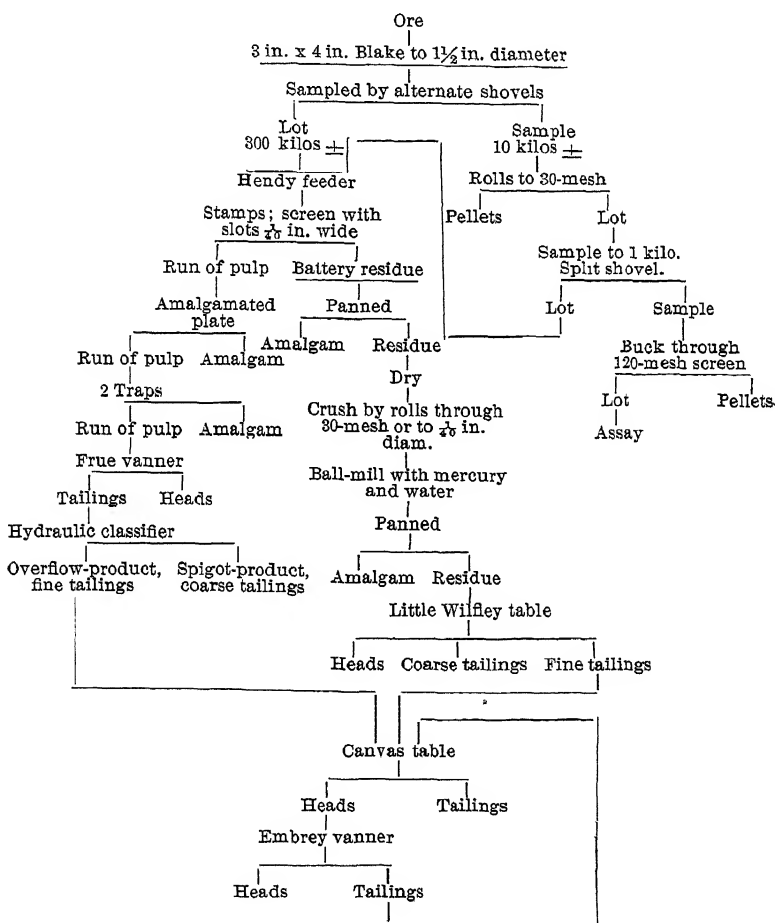
There are two periods of work in our laboratory; the fall term, when the class collectively goes through all our chief laboratory-processes, and the spring term, when the students work up their theses, each electing some one process or investigation and devoting himself wholly, or almost wholly, to that.

Some of the tests, the results of which are recorded in this paper, were made in the fall term; the preparation of the apparatus, the sampling and the assaying being done largely by assistants, with the aid of the students. Others were made in the spring term as thesis-work, and the results are due wholly to the students.

The complete "tree" of our process is patterned after the

most approved California mill-practice. It is modified only where the small-scale method must differ from the large, in order to secure results from a brief test lasting a few hours, followed, perhaps, by an interval of many months between tests, —which seek to rank in accuracy with those of the large-scale work, running over months and years of continuous work, broken only by brief intervals for cleaning-up.

### *Tree of Process.*



We use no battery-plates, because we believe the loss due to absorption of gold by the copper may be greater than the gain from using them.

We use silver amalgam on our apron-plate, painting it on

before the run, and scraping it off with the gold-contents afterwards, for two reasons: It renders the plate very quick to catch gold, and it renders the absorption-loss by the copper-plate insignificant, or prevents it altogether.

Our canvas and fine vanner-plant for treating the extremely fine tailings is that known as the Gates canvas-plant, originally designed by G. G. Gates and installed at the Kennedy mine, Jackson, California. Since it has been demonstrated clearly that this canvas-plant can pay only when the assay of concentrates is high, or when the percentage of concentrates is large, we do not always choose this run as the one to illustrate to students the wonderful catching-power of canvas for extremely fine concentrates. We do, however, make it our practice to separate, by classifying, the coarse Frue-vanner tailings from the fine, thereby preparing for canvas should we decide to use it.

*Sampling.*—Many of the ores we run would come under the head of "rich spotty" ores, as the Colorado phrase is. To crush and cut down such an ore, according to best Colorado rules of sampling, would require breaking the whole batch of from 500 to 1,000 lb. down to 10 or even 30-mesh. This would ruin our gold stamp-mill run. We are thus put in the odd position of having to choose between sampling the lot according to the best rules for safe work and spoiling the stamp-run, or of making a stamp-mill run, using only a very crude sample for valuation, which we could not possibly defend on the score of accuracy. It is needless to say we choose the latter method.

The mill and vanner were run under the following conditions:—

Apron-plate of copper 6 ft. long, 2 ft. wide, covered with silver amalgam; number of stamps, 3; weight of stamps, 225 lb.; diameter of shoe, 5.75 in.; number of drops, 98 per min.; height of drop, from 4.25 to 5.75 in.; height of discharge, 4.75 in.; screen, horizontally slotted, openings  $\frac{1}{16}$  in. wide; rate of crushing, 57 to 94 kilos. per hour; plate-slope, 1.5 in. per ft.; feed-water to the battery, 15 kilos. per min.; wash-water on vanner, from 4 to 8 kilos. per min.; Frue vanner-slope, 3.5 to 4.5 in. in 12 ft.; number of shakes, 198 per min.; belt-travel, from 40 to 60 in. per min.; canvas-table, 10 ft. long, 4 ft. wide; canvas-table slope, 1.5 in. per ft.; canvas-table feed, 10 kilos. water per min. and from 0.7 to 0.8 kilos. dry ore per min.; Em-

brey vanner, 1.5 in. per ft. slope, 80 in. per min. travel of belt, 240 vibrations of 0.5 in. per min.

The mill is a little 3-stamp Fraser and Chalmers prospecting-mill of 1882. It is very satisfactory in all respects except the number of stamps. A school-battery should have either two or five stamps; the former is preferred by us. Three stamps cannot have an order of drop which satisfactorily distributes the sand.

The vanner is a standard 4-ft. Frue, made by Fraser and Chalmers in 1882. It is a very good one; the belt being today as good as new. It is run from 5 to 10 days in a year.

The canvas-table is supplied with an accurate distributor. The canvas is No. 6 cotton duck, with the warp laid across the table for the greatest roughness, and is cut from a roll woven 10 ft. wide. The feed-water is the quantity, and the pulp is two to three times the quantity used by the Gates canvas-plant. The copper-plate is  $\frac{1}{8}$  in. thick, of softest copper.

The amalgam is made by dissolving 95 grammes silver in 380 c.c. nitric acid (1.2 sp. gr.) with addition of 817 c.c. water after solution. Pure mercury is then added at the rate of 16 parts mercury to 1 part silver. It should be stirred at short intervals during the formation of the amalgam. The result is a soft amalgam, clean and free from crystals or mercurous nitrate. Failure to follow this rule exactly is liable to give a very impure amalgam, which will require time and trouble to clean.

The apron-plate is cleaned preparatory to the application of silver amalgam as follows: It is washed off with water, scoured wet with infusorial earth or very fine sand, and then washed off with water again. Obstinate black spots can be removed by scouring wet with a flat piece of pumice-stone, adding mercury to plate the cleaned copper. It is then washed all over with a strong solution of cyanide, adding mercury, again rinsed off with water, and then rubbed dry with clean cloth or cotton-waste which must be free from oil. A moderate quantity of mercury is now rubbed on without water, after which the plate is ready for the amalgam. No water is allowed upon it yet.

The clean amalgam is squeezed in chamois skin to a moderately hard ball, and rubbed without water on the clean, bright, amalgamated copper-surface of the apron-plate. (A flat, bristle paste-brush may be useful to obtain an even distribution.) This coating is quick to catch gold, and prevents the gold from

coming in contact with the copper. No water is allowed upon it until stamping begins.

After the run, the plate is rinsed off with water, and the amalgam scraped off as clean as possible with a rubber-scraper. More mercury is rubbed on, and the plate is again scraped. This is repeated twice more, and all the amalgam is put together. If there is any hard amalgam which has not been removed, it is gently rubbed wet with a flat piece of pumice-stone in the presence of mercury, and so removed without injury to the plate. The amalgam from the plate, combined with that from the traps, is retorted; the residue is melted with borax in a crucible, and the bullion is parted for gold.

The battery-residue is worked up, as shown in the tree of the process, yielding amalgam, Wilfley heads, coarse tailings and fine tailings, which are added to the main portions of these products.

Since the battery is fed with pure mercury, while the apron-plate is coated with silver amalgam, the battery-amalgam enables us to get a valuation on the fineness of the gold caught at this point. We have no means to get this ratio on the portion of the gold caught on the apron-plate.

The mercurous nitrate and a little residual silver nitrate from making silver amalgam are worked over for mercury and silver by immersing strips of iron in the liquid, which precipitates the metals as amalgam.

The results of seven tests are here given :

*Ore No. 2423, from Nova Scotia, November 12, 1901.*

Class-work.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	384.8	1.03	0.003531	13.5873	.....
<i>Products:</i>					
Battery- and ball-mill-gold.....	.....	.....	.....	14.1545	87.40
Plate- and trap-gold.....	.....	.....	.....	0.6786	4.19
Vanner and Wilfley heads.....	19.9	0.84	0.002868	0.5708	3.53
Vanner and Wilfley coarse tailings.....	225.2	0.06	0.000213	0.4794	2.96
Vanner and Wilfley fine tail- ings.....	139.0	0.06	0.000206	0.2860	1.77
Unaccounted for.....	0.7	*1.03	0.003531	0.0247	0.15
Total.....	384.8	.....	.....	16.1940	100.00

\* Assumed for purposes of calculation.

Per cent. of gold-contents saved by amalgamation,	91.59
Per cent. of gold-contents in vanner concentrates,	3.53
Total,	95.12
	Oz. per Ton.
Assay by fire,	1.03
Assay based on mill-run,	1.23

*Ore No. 2425, from Nova Scotia, November 19, 1901.*  
Class-work.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	266.1	0.06	0.0002057	0.5475	.....
<i>Products:</i>					
Battery- and ball-mill-gold....	.....	.....	.....	0.03966	5.40
Plate- and trap-gold.....	.....	.....	.....	0.27700	37.72
Vanner and Wilfley heads....	11.6	0.06	0.0001936	0.02246	3.06
Vanner and Wilfley coarse tailings.....	178.9	0.05	0.0001695	0.30326	41.34
Vanner and Wilfley fine tail- ings.....	62.1	0.03	0.0001628	0.06387	8.70
Unaccounted for.....	13.5	*0.06	0.0002057	0.02777	3.78
Total.....	266.1	.....	.....	0.73402	100.00

\* Assumed for purposes of calculation.

Per cent. of gold-contents saved by amalgamation,	43.12
Per cent. of gold-contents in vanner concentrates,	3.06
Total,	46.18
	Oz. per Ton.
Assay by fire,	0.06
Assay based on mill-run,	0.08

*Ore No. 2444, from Nova Scotia, November 4, 1902.*  
Class-work.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	298.2	0.16	0.0005486	1.636	.....
<i>Products:</i>					
Battery- and ball-mill-gold....	.....	.....	.....	2.0300	78.43
Plate- and trap-gold.....	.....	.....	.....	0.1481	5.72
Vanner and Wilfley heads....	13.8	0.42	0.001440	0.1987	7.68
Vanner and Wilfley coarse tailings.....	186.7	0.02	0.00006857	0.1280	4.94
Vanner and Wilfley fine tail- ings.....	81.6	0.025	0.00008570	0.0699	2.70
Unaccounted for.....	16.1	*0.025	0.00008570	0.0138	0.53
Total.....	298.2	.....	.....	2.5885	100.00

\* Assumed for purposes of calculation.



Per cent. of gold-contents saved by amalgamation,	. . . . .	84.15
Per cent. of gold-contents in vanner concentrates,	. . . . .	<u>7.68</u>
Total,	. . . . .	91.83
		Oz. per Ton.
Assay by fire,	. . . . .	0.16
Assay based on mill-run,	. . . . .	0.253

*Ore No. 2444, from Nova Scotia, November 11, 1902.*  
Class-work.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	338.8	0.15	0.0005142	1.742	.....
<i>Products:</i>					
Battery- and ball-mill-gold....	.....	.....	.....	1.9969	75.16
Plate- and trap-gold.....	.....	.....	.....	0.2022	7.61
Vanner and Wilfley heads....	15.0	0.42	0.001440	0.2160	8.12
Vanner and Wilfley coarse tailings.....	205.2	0.02	0.00006857	0.1407	5.29
Vanner and Wilfley fine tail- ings.....	107.4	0.025	0.00008570	0.0921	3.46
Unaccounted for.....	11.2	*0.025	0.00008570	0.0096	0.36
Total.....	338.8	.....	.....	2.6575	100.00

\* Assumed for purposes of calculation.

Per cent. of gold-contents saved by amalgamation,	. . . . .	82.77
Per cent. of gold-contents in vanner concentrates,	. . . . .	<u>8.12</u>
Total,	. . . . .	90.89
		Oz. per Ton.
Assay by fire,	. . . . .	0.15
Assay based on mill-run,	. . . . .	0.229

*Ore No. 2444, from Nova Scotia, November 18, 1902.*  
Class-work.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	332.6	0.25	0.0008570	2.850	.....
<i>Products:</i>					
Battery- and ball-mill-gold....	.....	.....	.....	1.9820	76.44
Plate- and trap gold.....	.....	.....	.....	0.0850	3.28
Vanner and Wilfley heads....	12.9	0.40	0.001373	0.1769	6.82
Vanner and Wilfley coarse tailings.....	201.2	0.03	0.0001028	0.2069	7.98
Vanner and Wilfley fine tail- ings.....	114.1	0.035	0.0001200	0.1370	5.28
Unaccounted for.....	4.4	*0.035	0.0001200	0.0053	0.20
Total.....	332.6	.....	.....	2.5931	100.00

\* Assumed for purposes of calculation.

Per cent. of gold-contents saved by amalgamation, . . .	79.72
Per cent. of gold-contents in vanner concentrates, . . .	6.82
Total, . . . . .	86.54
	Oz. per Ton.
Assay by fire, . . . . .	0.25
Assay based on mill-run, . . . . .	0.227

*Ore No. 2444, from Nova Scotia, February, 1902.*

Thesis of C. H. Burr and B. E. McKechnie.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	356.2	0.13	.....	.....	.....
<i>Products:</i>					
Battery-gold.....	.....	.....	.....	2.01902	72.53
Plate-gold.....	.....	.....	.....	0.22736	8.17
Vanner heads.....	16.8	0.44	0.001511	0.25380	9.39
Vanner coarse tailings.....	208.6	0.02	0.00007401	0.15444	5.54
Vanner fine tailings.....	122.3	0.03	0.0009900	0.12107	4.35
Unaccounted for.....	8.5	*0.02	0.00007	0.00566	0.20
Total.....	356.2	.....	.....	2.78135	100.18

\* Assay assumed for purposes of calculation.

Per cent. of gold-contents saved by amalgamation, . . .	80.70
Per cent. of gold-contents in vanner concentrates, . . .	9.39
Total, . . . . .	90.09
	Oz. per Ton.
Assay of ore by fire, . . . . .	0.13
Assay based on mill-run, . . . . .	0.222

*Ore No. 2361, from Nova Scotia, March, 1901.*

Thesis of Henry C. Marcus.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed.....	256.30	0.13	0.000444	1.13797	.....
<i>Products:</i>					
Battery-gold.....	.....	.....	.....	2.38216	74.84
Plate gold.....	.....	.....	.....	0.51440	16.16
Vanner heads.....	15.65	0.11	0.000377	0.05904	1.86
Vanner coarse tailings.....	195.85	0.02	0.000069	0.13156	4.13
Vanner fine tailings.....	43.45	0.057	0.000195	0.08493	2.67
Unaccounted for.....	5.35	*0.057	0.000195	0.01043	0.33
Total.....	256.30	.....	.....	3.18252	99.99

\* Assumed for purposes of calculation.

Per cent. of gold-contents saved by amalgamation, . . . . .	91.00
Per cent. of gold-contents in vanner heads, . . . . .	1.86
Total, . . . . .	92.86
	Oz. per Ton.
Assay of ore by fire, . . . . .	0.13
Assay based on mill-run, . . . . .	0.36

*Summary of the End-Results of the Seven Tests.*

Ore No.	PERCENTAGE OF GOLD IN CRUDE ORE.						Assay by Fire, Oz. per Ton.	Assay by Mill-Test, Oz. per Ton.
	Battery- and Ball-Mill.	Plate and Trap.	Vanner and Wilfley Heads.	Vanner and Wilfley Coarse Tailings.	Vanner and Wilfley Fine Tailings.	Unaccounted for.		
2423	87.40	4.19	3.53	2.96	1.77	0.15	1.03	1.23
2425	5.40	37.72	3.06	41.34	8.70	3.78	0.06	0.08
2444	78.43	5.72	7.68	4.94	2.70	0.53	0.16	0.253
2444	75.16	7.61	8.12	5.29	3.46	0.36	0.15	0.229
2444	76.44	3.28	6.82	7.98	5.28	0.20	0.25	0.227
2444	72.53	8.17	9.39	5.54	4.35	0.20	0.13	0.222
2361	74.84	16.16	1.86	4.13	2.67	0.33	0.13	0.36

Attention is called to the slight variation in the mill-assay of the four tests on No. 2444, and also to the considerable variation in the assays by fire.

*Remarks.*—The method here used in tabulating the results of a test has been found to be a most concise and satisfactory one. As the value of the ore obtained by the mill-run is much more accurate than that based on the assay of the ore, the percentages found in the column headed "Per cent. of the total gold" are based on the total gold in the ore as found by the mill-run, and not on the assay of the feed; in fact, one of the objects of this work is to show the student the impossibility of valuing a free-milling gold-ore by sampling and assaying.

The degree of accuracy attained here is, we believe, up to that of the very best mill-practice. It has been attained by locating the causes of error, and then by removing them as completely as possible, even by departing, if necessary, from large-scale mill-methods.

## Electrical Power-Transmission for Mines.

BY FRANCIS O. BLACKWELL, SCHENECTADY, N. Y.

(Albany Meeting, February, 1908.)

THERE are few industries in which power is more important to successful operation than mining, and none in which it is so difficult to obtain power cheaply.

Fuel is usually expensive in mining districts, and the machinery may be widely scattered and located in places difficult of access. The work is often so intermittent and variable as to prevent the economical use of steam. Mining operations are likely to exhibit accumulations of comparatively small and inefficient engines, numerous boilers and long lines of wasteful steam-pipes.

Compressed air, as a means of transmitting power, is so handicapped by losses in the compressors and piping that it is limited to short distances and to the operation of apparatus requiring power, without regard to cost.

To-day, electricity is rapidly replacing other methods of power-distribution, primarily because it is practically independent of distance. By selecting a potential proportional to the length of transmission, the cost of electric conductors and the efficiency are the same for all distances within reasonable limits. For instance, a given amount of power can be delivered either 528 ft. away at 100 volts, or 50 miles at 50,000 volts, with the same total of copper per H.P. and with exactly the same loss of power in the transmission-circuit. This independence of distance permits the selection of the place most advantageous for power-development as a site for an electric power-plant, and the concentration there of the generation of all the power required within a radius of many miles. The power-site may be at a fall of water, if one is available, or a steam-plant can be built where fuel and supplies are cheapest, or where sufficient water is available for boiler-feed and condensing purposes.

The concentration of power in one place is in itself an evident economy, even if there are no other advantages to be obtained by the use of electricity. The first cost of one large unit is, of course, less than that of several smaller ones, and a large engine is inherently more economical in the use of steam and cost of labor. In addition, the large plant is naturally laid out with many auxiliaries for reducing the expense of operation that would not be considered in smaller installations. Every steam-engine has a load at which it runs with the highest economy; at greater or less loads its rate of steam-consumption increases.

In mines where separate engines are used for different purposes, it is exceptional to find one that is running at its proper load. In the electric central station, however, the sizes of the steam-units may be so chosen that they can always be operated at approximately their most efficient loads, by shutting down or starting up engines as required.

Where many fluctuating and intermittent loads are thrown together on one plant, they average up and become a steady load. An electric railway-plant is the best example of this. The motors on each car are constantly taking different amounts of power, yet there is no appreciable variation in the output of the station; furthermore, the station-capacity need be only large enough for the average power. A railway power-house of large capacity seldom delivers more than one-tenth of the power which might be required if all the motors ran at their maximum capacity at the same instant. Even if there is but one fluctuating load, such as a large hoist, it is more economical to combine it with whatever steady load there may be, and so keep the engine loaded constantly up to an efficient point.

Electricity also permits the efficient division of power into large or small units, which require practically no attention and are suitable for every purpose. By a proper choice of potential and system, the transmission-losses need never exceed 10 per cent, even when power is carried 150 miles.

There is another feature in which the electric differs radically from other systems of power-transmission. With steam, air or mechanical methods the losses are nearly constant at all loads. The efficiency, therefore, at partial loads is lower than at full load. With electricity the efficiency is higher at par-

tial loads, and there is no power whatever lost in the transmission-system when no work is being done.

The steam-turbine, which promises to revolutionize the generation of power from coal, is only suited to driving electrical machinery on account of its high speed.

Electrical apparatus is the most efficient machinery known to engineering. Generators and motors, even in comparatively small sizes, have over 90 per cent efficiency, and large machines go up to 97 per cent. It is only necessary to compare this with water-wheels of 70 or 80 per cent efficiency, and steam-engines of 15 per cent heat-economy, to see the remarkable results that have been achieved in electrical design.

Electrical systems of distribution may be divided into three general classes, shown in the diagrams, Figs. 1, 2 and 3.

1. The generator delivers current to the distribution-system at low potential, and the generators are wound for just enough higher potential to take care of the losses in transmission.

2. The generator delivers power to the distribution-system at high potential, and step-down transformers are used to lower the pressure to that of the motors and lights.

3. The generator is wound for a low potential, and transformers are employed to raise the pressure to that required for economical transmission, and again to lower it for distribution.

A plant of the first kind is shown diagrammatically in Fig. 1. The generator is wound for any convenient potential from 100 to 600 volts, and may be either continuous or alternating. Motors and lights are operated from it directly. It is adapted to small or compact plants where the distance to which power is to be distributed is not great.

Fig. 2 shows a plant of the second class. The generator is wound for, say, 2,300 volts, and must be alternating. Large motors may be made for this potential, but lights and small motors must take their power through lowering transformers. Generators of large size can be wound for pressure up to 15,000 volts, so that this system covers the greater portion of the power-transmission field, and is that used most generally by lighting and power central stations where the distances are not over 15 miles.

When the distances over which power must be carried are so great as to require a higher potential than the generator can

be wound for, the system must be of the third class, as shown in Fig. 3. The generator is wound for any convenient low potential, and the pressure is raised by means of step-up transformers to a higher pressure, for instance, 20,000 to 50,000 volts. At the distribution-end the pressure is reduced to a potential suitable for use.

Fig. 3 shows two sub-stations. In one, the power is distributed at an intermediate potential, a second set of step-down transformers being employed to supply most of the motors and lights. In the other sub-station, the load is supplied with low-voltage current through a single bank of transformers. This type of plant is used for the longest transmissions, and is to-day commercially delivering power at a distance of 150 miles.

A power-system, of course, is not necessarily confined to one of the methods given. The same plant often combines two, and sometimes all three, kinds of distribution.

The continuous-current was the first electric system to be used, and is too well known to require description here.

The alternating-current was developed later, and has the advantage over the continuous of making possible the use of higher potentials. The simpler construction of alternating generators and motors, with stationary windings and without commutators, permits them to be safely wound for high pressures. Alternating current can be converted from one potential to another with the greatest facility, by means of transformers without moving parts.

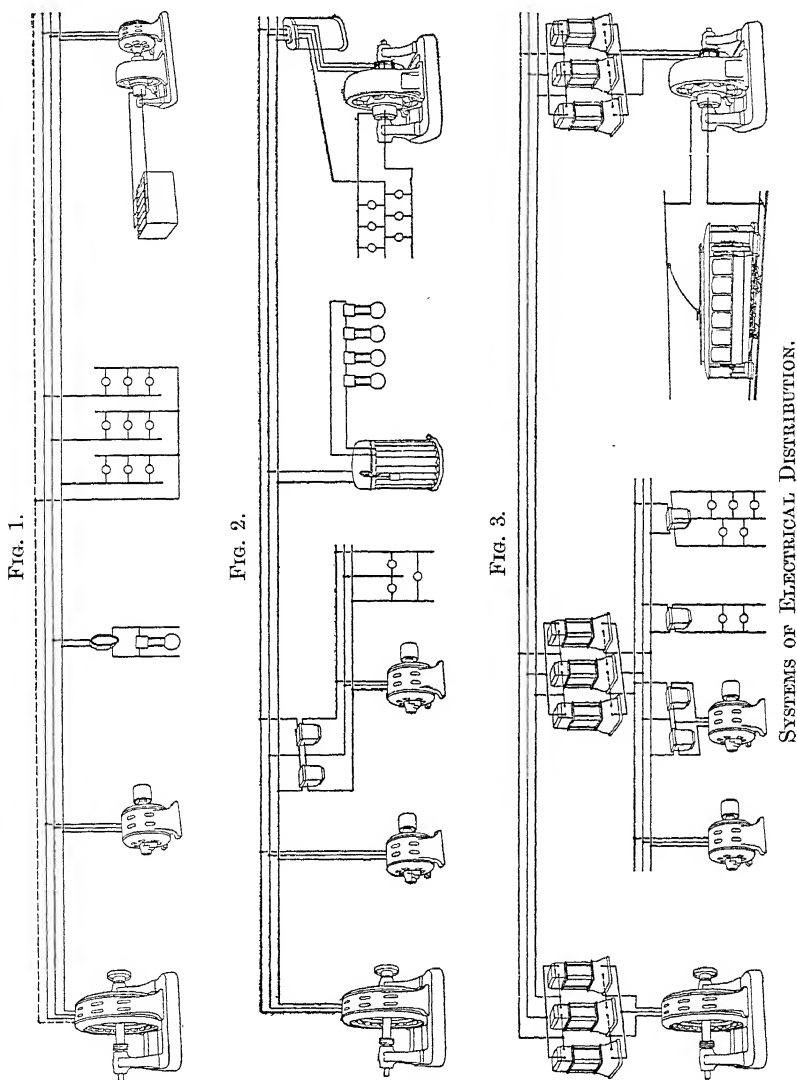
Alternating motors are of two kinds, synchronous and induction. The synchronous motor is exactly like the generator, and is suitable for large units of power, where constant speed is desired. The induction-motor permits variation in speed, and is particularly adapted to mining-work, on account of the absence of complication in its construction and operation.

Figs. 5 and 6 show induction-motors used for mining-purposes; and Fig. 7 the armature of one of them.

*Hoists.*—The economy of steam-hoisting is necessarily low. The intermittent service and the varying load are serious handicaps to even the most efficient type of engine; but, as a rule, little attempt is made to secure low steam-consumption.

To start the load, even by carrying steam to full stroke, a larger cylinder is required than can efficiently be used after the

load is in motion. This is due to the fact that one of the cranks may be dead-centered, so that the power applied through the other must be large enough to overcome the friction, always greatest at the instant of starting. At starting there is gener-



SYSTEMS OF ELECTRICAL DISTRIBUTION.

ally the weight of the hoisting-rope to be lifted and the weight of the entire system to be accelerated. Once up to speed, the requisite power is much reduced, and the engine is working at too low a load for steam-economy.



The throttling of steam to control speed; the necessity for reversing the engine; the variation in steam-pressure; the absence of condensing apparatus; the cooling and large clearance of cylinders, and the condensation and leakage of steam in pipes when doing no work, are all against the steam hoisting-engine. One of the largest hoisting-engines in the world was recently tested and found to take 60 lb. of steam per indicated H.P.-hour. The electric-motor, on the other hand, is ideal for intermittent work. It wastes absolutely no energy when at rest, there being no leakage or condensation. Its efficiency is high, from one-quarter load to twice full load.

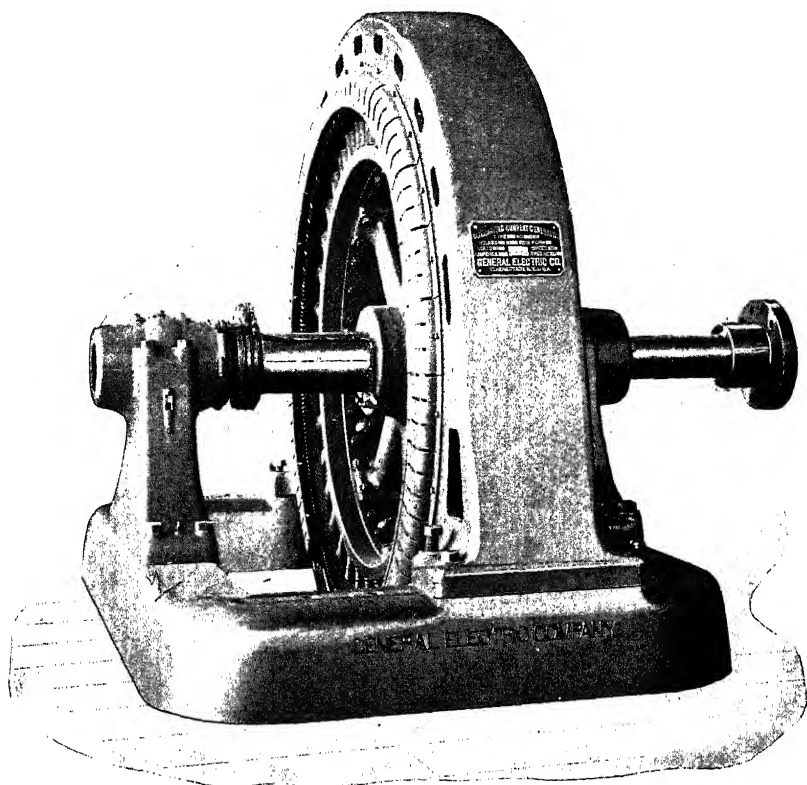
For an electric hoist, however, we must consider not the motor alone, but also the source of the electric power, and secure the most efficient conditions for the supply-plant.

The average H.P. taken by an electric hoist is seldom over 25 per cent, and often considerably below 10 per cent, of the maximum; and it is this low-load factor that has prevented, heretofore, the extensive use of motors on hoists of the largest size. The efficiency of a steam-hoist per H.P.-hour may be extremely low; but if we take the total steam consumed and assume that the hoist is operated by it at its maximum power continuously, we get a fair apparent economy.

An electric supply-plant must have sufficient capacity for the maximum demand upon it. If it is a water-power plant, in which the interest on the investment mainly determines the cost of power, an occasional high load costs nearly as much as if that load were run continuously. In a steam-plant, on the other hand, a variable load, if it is a large percentage of the total load, increases the fuel-consumption per H.P. It is, therefore, particularly desirable, in using an electric hoist, to keep the load as constant as possible and cut down the maximum.

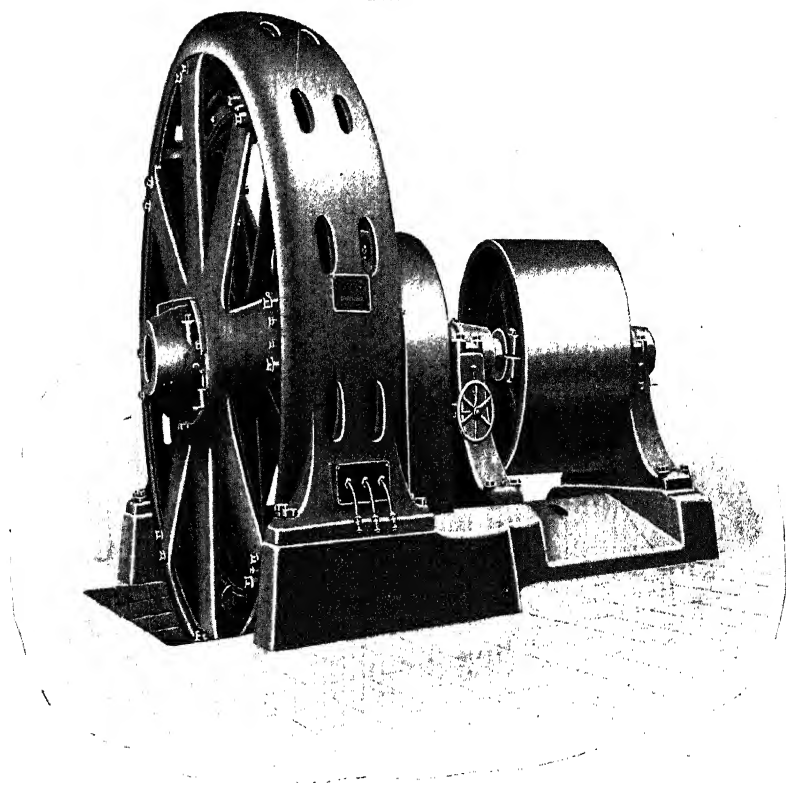
A mistake often made with such a hoist is that of running at an unnecessarily high hoisting-speed with too light a load of ore. The time lost in loading and unloading the skip or cage is the same for heavy as for light loads, so that the slower the hoisting-speed, the smaller is the proportion of the total time wasted at the ends of the run. With many a hoist, doubling the load and halving the speed would get out more ore in a day. This is advantageous for the motor, because it reduces the relative dead-weight, friction and rate of acceleration, and

FIG. 4.



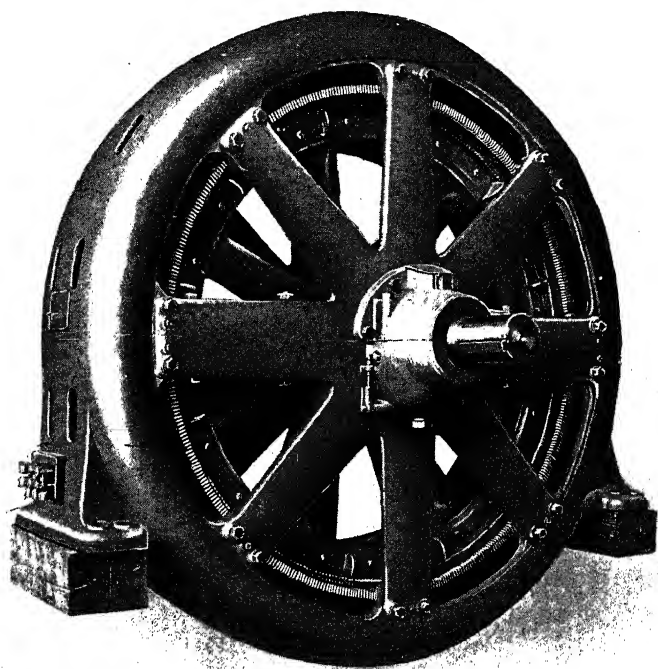
2,000-H.P. ALTERNATING-GENERATOR. WEST KOOTENAY L. & P. Co.,  
ROSSLAND, B. C.

FIG. 5.



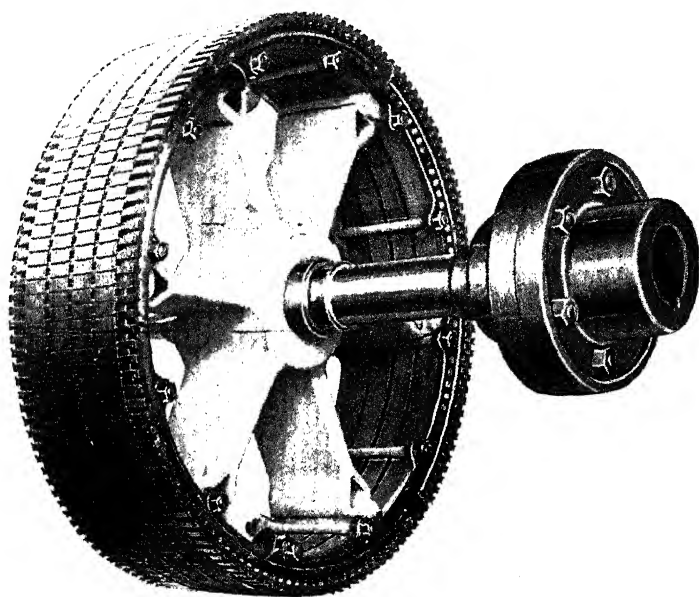
800-H.P. INDUCTION-MOTOR, DRIVING AIR-COMPRESSOR. ANACONDA MINING CO., BUTTE, MONT.

FIG. 6.



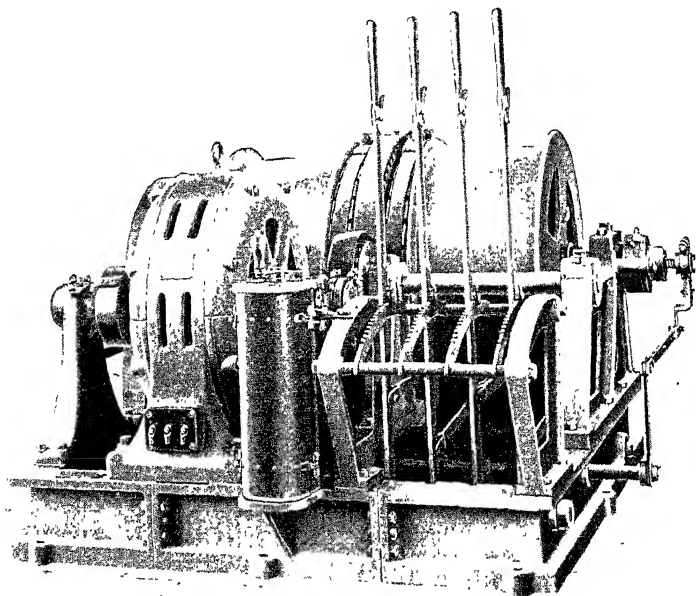
700-H.P. 150 R. P. M. INDUCTION-MOTOR, DRIVING SAND-WHEEL AT CALUMET  
& HECLA MINES.

FIG. 7.



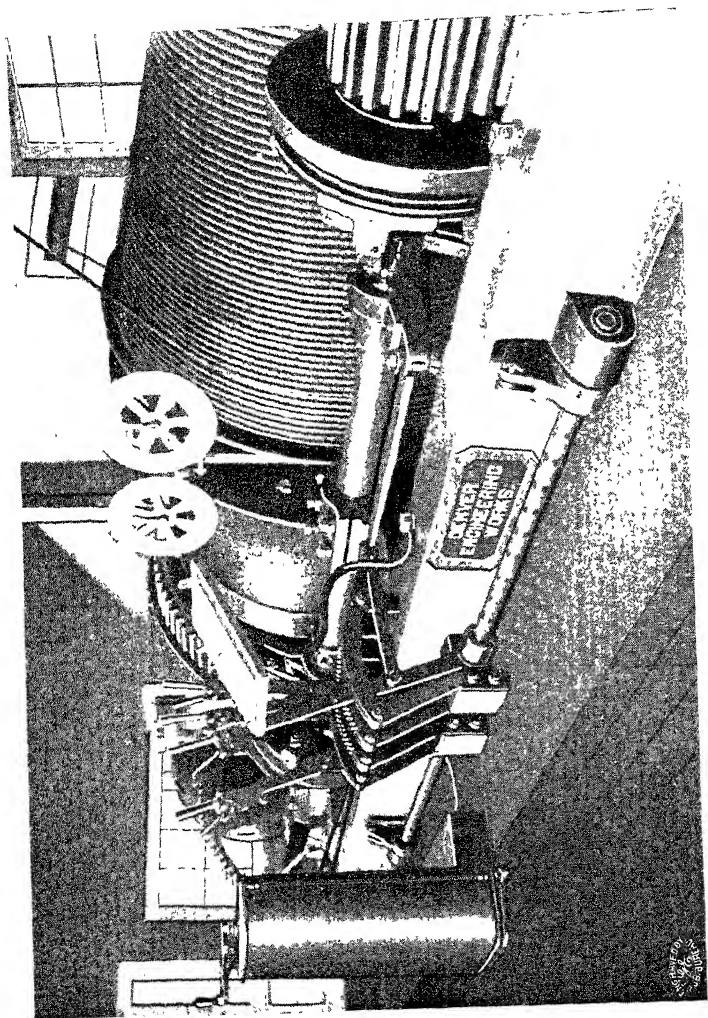
ARMATURE OF INDUCTION-MOTOR.

FIG. 8.



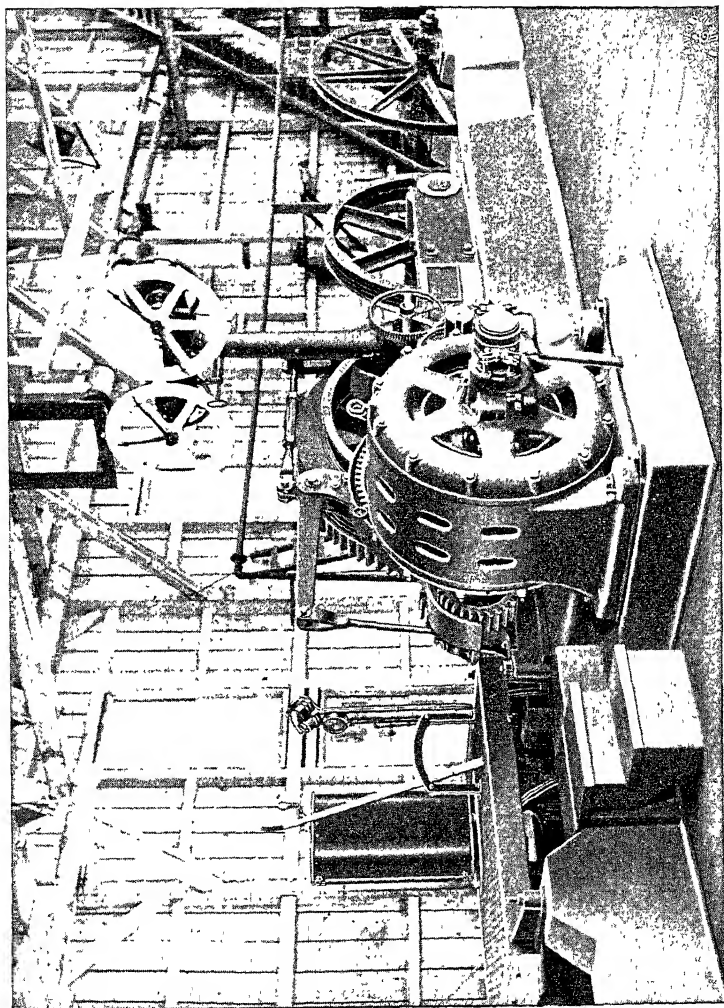
DOUBLE FRICTION-DRUM LIDGERWOOD HOIST, WITH INDUCTION-MOTOR, IN USE AT KOLAR GOLD-FIELDS, INDIA.

FIG. 9.



CONTAL, DOUBLE-DRUM ELECTRIC HOIST, AT JOSIE MINE, LE ROI No. 2 MINING Co., ROS-LAND, B. C.

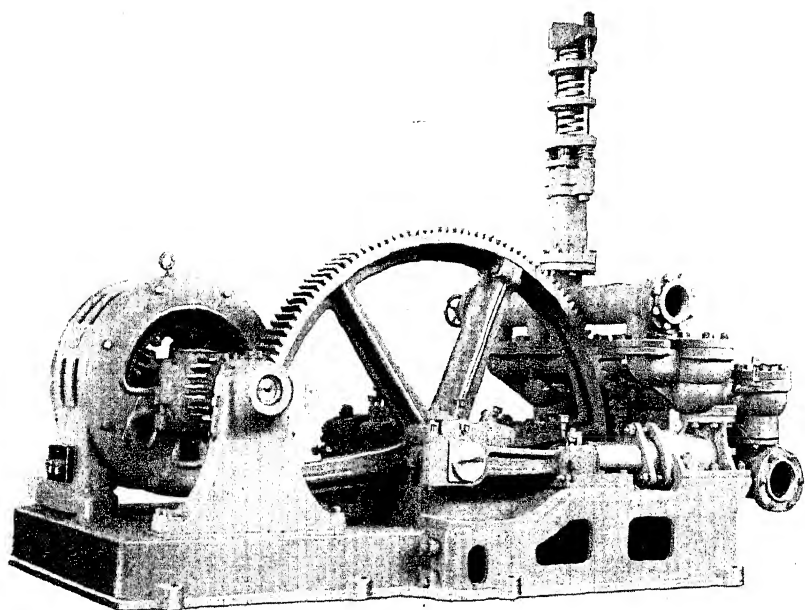
FIG. 10.



ELECTRIC HOIST AT UNION SHAFT, COMSTOCK MINES, NEVADA.

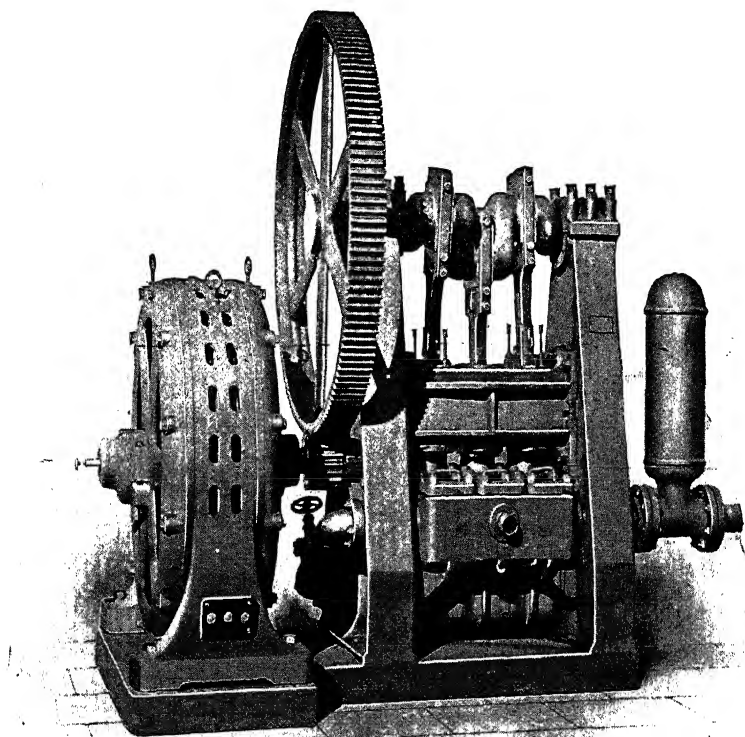


FIG. 11.



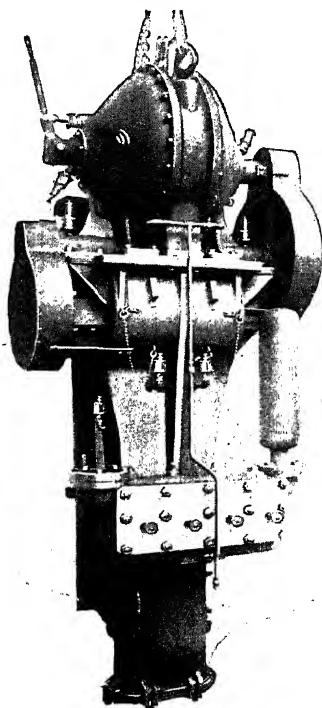
ELECTRIC PUMPS FOR 200 GALLONS A MINUTE AGAINST 500-FT. HEAD, FOR  
CONSOLIDATED GOLD-FIELDS, SOUTH AFRICA.

FIG. 12.



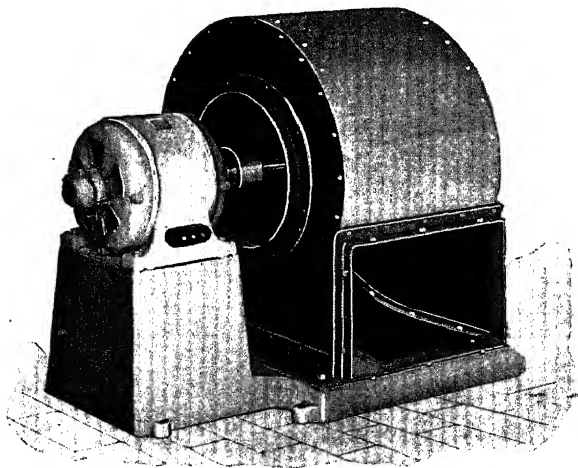
VERTICAL TRIPLEX PUMP, DRIVEN BY INDUCTION-MOTOR.

FIG. 13.



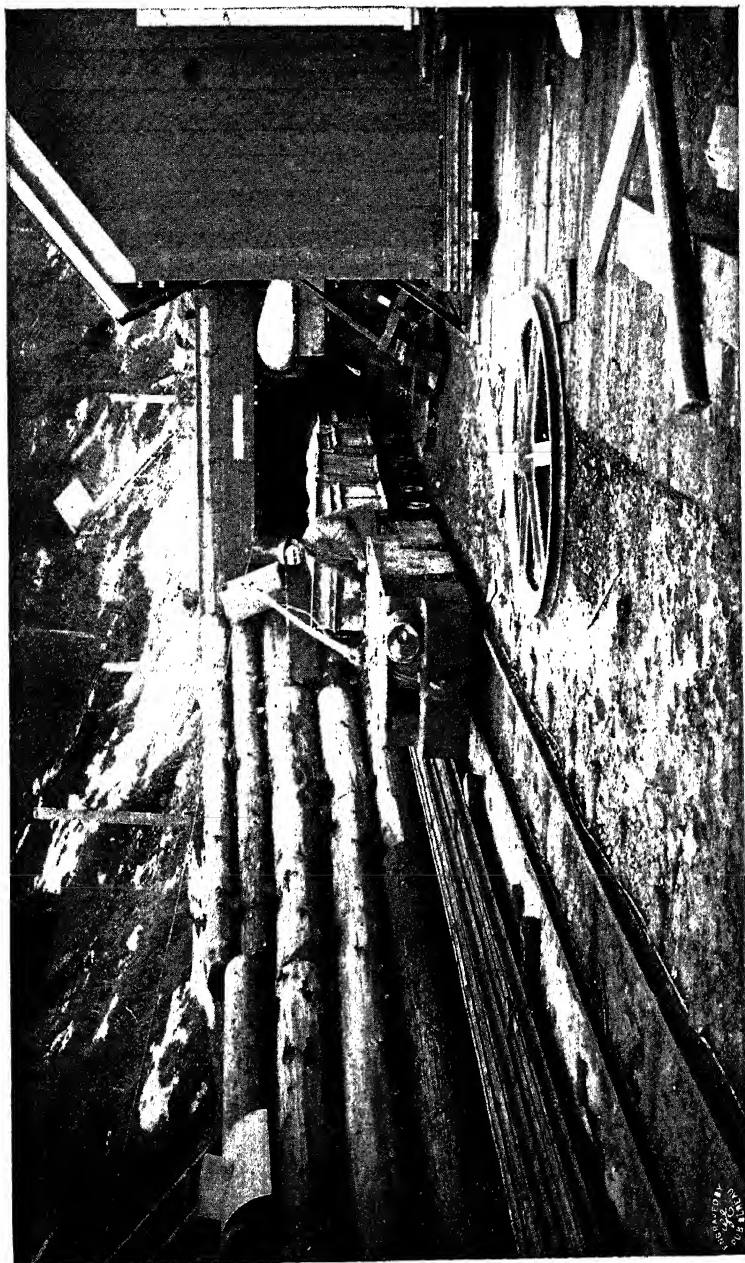
KNOWLES 6 x 6 DUPLEX SINKING-PUMP, WITH 20-H.P. WATERPROOF INDUCTION-MOTOR.

FIG. 14.



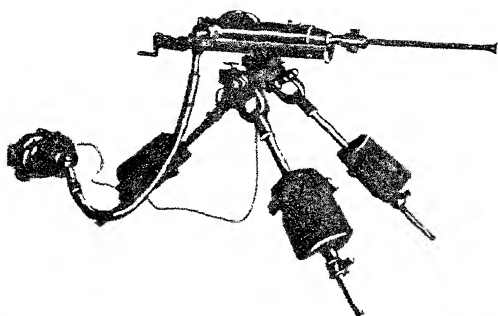
LOW-PRESSURE BLOWER, DRIVEN BY ELECTRIC-MOTOR.

FIG. 15.

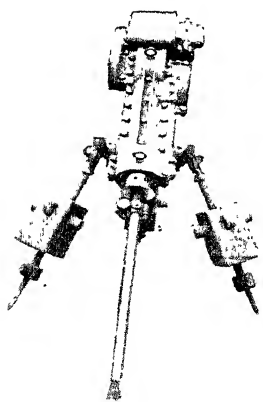


4.5-TON ELECTRIC MINE-LOCOMOTIVE, WITH TRAIN OF ORE-CARS AT MOUTH OF CROSS-CUT TUNNEL.

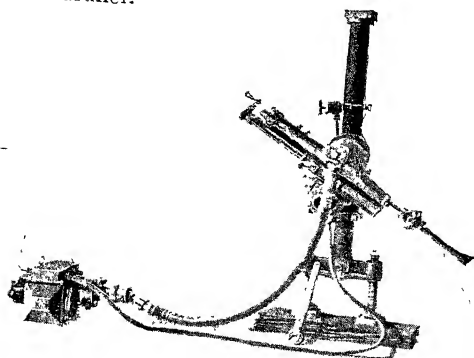
FIG 16.



Gardner.



Box.



Durkee.

ELECTRIC ROCK-DRILLS.

makes the load more uniform. Moreover, it contributes to the ease of handling, and reduces the wear on rope and machinery.

In steam-hoisting, the cost of power can be reduced by running in balance, especially if a conical drum, or a flat-rope reel, is also employed, to compensate for the weight of the hoisting-rope. For electric motors, these devices, which tend to keep the load more constant and reduce the maximum *torque* required at starting, are even more important.

The following are excellent examples of electric hoists of this kind:

*Flat-Rope Double-Reel Hoist at Free Silver-Mine, Aspen, Colorado.*

Designed by D. W. Brunton in 1895.

Diameter of reel, empty, 5 ft.; full, 10 ft.

Speed of hoisting ore, 20 rev.= 315 to 630 ft. per min.

Speed of bailing water, 32.6 rev.= 510 to 1020 ft. per min.

Rope, 1,500 ft. of 4- by  $\frac{3}{4}$ -in. flat steel-cable, weighing 2.75 lb. per ft.

Weight of cage, 1,375 lb.; bucket suspended from cage, 500 lb.;

car, 1,000 lb.; rock and ore in car and bucket, 3000 lb.; bailer,

1,950 lb.; water, 6930 lb.

Weight of counter-weight with cage and bucket, 2450 lb.

Weight of counter-weight with cage and car, 2675 lb.

Weight of counter-weight with bailer, 5126 lb.

Motor, "General Electric," 120-H.P., 650 rev. per min., 500 volt, continuous current. Different gears are used to give 20 and 32.6 rev. to the drum.

Shaft, vertical single-compartment, with guides for counter-weight.

The following test was made in hoisting from 835-ft. level with bailer and its counter-weight, with 20-rev. gear:

Current at 525 volts, taken at 10-sec. intervals.

*Hoisting*: 90, 80, 80, 70, 70, 70, 70, 70, 70, 70, 75, 75, 75 amperes.

Average, 74 amp.= 39 K.W.= 52 H.P.

Average theoretical power, 42 H.P.; efficiency, 80 per cent.

*Lowering*: 90, 90, 90, 70, 70, 70, 70, 60, 60, 58, 57, 55, 50 amperes.

Average, 68 amp.= 36 K.W.= 47 H.P.

Average theoretical power, 36 H.P.; efficiency, 76 per cent.

If this had been a single flat drum-hoist, without counter-weight, it would, probably, have taken 350 H.P. at starting and 200 when up to speed. The maximum power is, therefore, reduced to about one-fifth by over-counterbalancing and by the variation in the diameter of the reel through the piling-up of the rope on itself. Although a single-compartment shaft, the work of hoisting is divided between the hoisting and lowering.

*Conical Double-Drum-Hoist at Josie, Le Roi No. 2 Mining Co., Rossland, B. C.*

Designed by D. W. Brunton and built by the Denver Engineering-Works in 1899.

Diameter of drums, 4 ft. 10 in. minimum to 7 ft. 4 in. maximum.

Face, 5 ft. Speed, 32 rev. = 484 to 732 ft. per min.

Rope, 1,000 ft. of  $\frac{3}{8}$ -in. steel, weighing 1.24 lb. per ft.

Weight of cage, 2,800 lb.; car, 700 lb.; ore, 2,000 lb.

Motor, "General Electric," three-phase induction, 150 H.P., 600 rev. per min., 220 volts, with secondary rheostatic control.

Shaft inclined  $72^\circ$  from horizontal (nat. sin.  $72^\circ = 0.95$ ).

Observations of current at 240 volts, made by Mr. L. A. Campbell when hoisting from 700-ft. level.

In Balance.		Starting.		Full Speed.		
Trip No.	Amperes.	H.P.	Amperes.	H.P.		
1.....	300	141	225	106	} Theoretical power at starting, 30 H.P., and at full speed, 40 H.P.; efficiency, 40 per cent.	
2.....	225	106	220	104		
3.....	220	104	225	106		
4.....	225	106	230	108		
5.....	230	108	225	106		
Out of Balance.						} Theoretical power at. starting, 100 H.P., and at full speed, 111 H.P.; efficiency, 72 per cent.
6.....	300	141	315	148		
7.....	315	148	315	148		
8.....	325	153	340	160		

The efficiency in this case is lower than it would be with a vertical shaft. It will be noticed, however, that the current at starting is no greater than at the end of the hoist, which is due to the fact that the drums are coned more than is necessary to balance the weight of the rope. The starting-current is probably not over one-half what it would have been with flat drums, either in or out of balance. This hoist is illustrated in Fig 9. Three of these hoists have been installed in Rossland, and have been very satisfactory. Power is obtained from the West Kootenay Power and Light Co., the water-power plant of which is at Bonnington Falls, 32 miles from Rossland. About 4,000 H.P. in all is used for mining purposes, and I am informed that there is no difficulty in maintaining both speed and potential regulation constant within 5 per cent.

Another type of electric hoist is being extensively installed on the Comstock Lode by Mr. Leon M. Hall. These hoists are operated on the tail-rope system, so that all weights are balanced except that of the ore itself. They run at speeds up to 1,200 ft. per min. with unbalanced loads of from 3,200 to 3,800 lb., and from depths of from 850 to 2,500 ft. The motors

are 3-phase induction-motors of 75, 100 and 200 H.P. Power is supplied by the Truckee River General Electric Co. from a water-power, and over a 35-mile line to Virginia City. The hoist installed at the Union Shaft is shown in Fig. 10.

These hoists are most successful and economical in operation. The tail-rope system is not applicable to all mines, because it does not permit frequent changes, so as to hoist from different levels, or the use of one compartment of the shaft independently of the other, or over-counterbalancing, so as to compensate for the acceleration at starting, as can be done with conical drums.

The Metropolitan Coal Co. of Boston, Mass., has recently installed an electric coal-hoist, having a number of interesting features which I believe could be advantageously employed in mining-work.

There is a large coal storage-pile on the Company's wharf; and the hoist is used to unload vessels and to transfer coal from the storage-pile to the boilers of the Edison Illuminating Co.'s station, or to bins for distribution in the city by wagon.

The bucket is of the automatic clam-shell type, weighing with load about 8,000 lb., and the lift is 60 ft. The hoist has two 3-ft. drums directly connected to two continuous-current motors, rated at 100 H.P. each and running at 150 rev. per min.

The operator is located at the top of the tower, out of sight of the hoist, where he can have an unobstructed view of the bucket. Both hands are occupied with the levers of the clutches which throw in the drums. The power and speed must, therefore, be governed by a controller, operated by a few inches' movement of his foot.

The large current required, amounting to over 1,000 amperes, at 250 volts, made it impracticable to use any ordinary form of controller, because it would be too hard to move. A special type of controller was, therefore, contrived, consisting of large switches, moved by electric magnets placed near the motors.

A small master-controller connected to the magnets by fine wires throws the switches into any desired combination. The master-controller can be located at any distance from the hoist, and requires almost no power to move it.

The hoisting-speed of 1,200 ft. per min. is reached in 1 or 2 sec. (an extraordinary acceleration), and the hoist has made



4 round trips per min. If this rate were kept up continuously for 24 hours, the machine would hoist 9,500 tons of coal a day. Actually, it does not do anything like this amount, as the tower has to be moved so as to be over new hatchways, and much time is lost in changing, and in other ways.

Fig. 8 illustrates one of the common types of small electric hoists.

*Pumps.*—To use, for pumping, steam or air with any degree of expansion requires a fly-wheel; otherwise the steam- or air-consumption will be large, even when water is available for condensing. For such pumping-engines, located underground and distant from the boiler- or compressor-plant, the transmission-efficiency must be low. The electric-system, on the other hand, delivers power to the pump with little loss. The efficiency of even small electric mine-pumps is not far from 80 per cent, and large pumps have been reported to give as high as 90 per cent.

To regulate properly the speed of motors, so as to vary the amount of water pumped, has been a serious problem. With the continuous current, control by field-variation, as now generally employed for the electrically-equipped tools of machine-shops, is most successful. The electric-pumps running for 10 years past in the Calumet and Hecla mine, Michigan, have a two-to-one adjustment of speed by this system. With the alternating-current motor, field-control is impracticable, and other methods must be employed.

The windings of the induction-motors can, however, be thrown into different combinations for various numbers of poles and speeds, and several frequencies can also be provided for.

The high-head centrifugal pumps now under construction in Europe and this country, having a number of runners in series, will probably, in time, take the place of reciprocating-types. The high speed and rotary motion of these pumps are perfectly adapted to direct connection with electric motors; and as they are devoid of gearing, valves and reciprocating parts, they run without noise or vibration.

Instead of a speed of 25 or 50 strokes per min., the high-head centrifugal pump will run at 1,000 or 2,000 rev. per min., and the weight and size of both motor and pump will be but a

fraction of those of the corresponding reciprocating-pumps. It has also the advantage of requiring little or no speed-variation. A change in speed of, say, 20 per cent will make all the difference between delivering no water and the full capacity. If either the suction- or the discharge-pipe is closed, there is no great increase in pressure; and the pump unloads itself like a blower. The capacity, therefore, may be regulated by simply opening and closing a valve. Any considerable variation in the suction-head will make the pump automatic. Two well-designed types of reciprocating-pumps, about 50 of which are in operation in the mines near Johannesburg, South Africa, are shown in Figs. 11 and 12. Fig. 13 shows a sinking-pump in which the motor is entirely enclosed and water-proof.

*Air-Compressors.*—Air-compressors are another class of machines in which the speed would naturally be regulated to compensate for changes in load. The speed of the motors driving them can be varied, as is done with pumps; but it has generally been found preferable to control the discharge of air in other ways.

By providing a by-pass between the end of the cylinders, the air can be made to circulate freely from one side of the piston to the other, thus unloading the compressor (except as to friction), and delivering no air into the receiver. The objection to this method is that it is constantly jumping from no load to full load, which is an undesirable condition for the power-station.

Another way is to throttle the air-intake, which enables the compressor to run continuously at partial loads. Neither of these methods is as efficient as that employed on the Nordberg compressor, in which there are positively-operated suction-valves, which permit the cylinder to fill with air, more or less of which is freely ejected on the return-stroke, before the valves close. The air-pressure in the receiver automatically controls the cut-off of the valves, and, consequently, the amount of free air compressed at each stroke.

A compressor of this type, driven by a 300-H.P., 3-phase 25-cycle induction-motor, is in process of installation at the Witherbee-Sherman mines near Port Henry, N. Y. This motor can be run at half-speed with full efficiency.

*Stamp-Mills.*—The operation of stamp-mills by electric-motors

has the advantage of doing away with nearly all shafting and belting and the losses incident thereto; each piece of machinery being directly driven by its own motor. This arrangement, besides being more efficient, makes it possible to shut down any part of the mill and run any other part.

The close speed-regulation of the electric-motor allows concentrating- and similar apparatus to be run up to the limit of speed, and this naturally increases the output over the variable speeds usual with belt-driven apparatus.

Rock-crushers and grinding-rolls, on the other hand, require a variable speed, in order that the fly-wheels may give up their stored energy to meet overloads. Separate electric-motors which slow down under load, however, accomplish this result perfectly, and, at the same time, prevent the crushing-machinery from affecting the speed-regulation of the concentrators.

The new mills of the Calumet and Hecla Co. at Lake Linden, Mich., are good examples of this kind of equipment. They have two 25-cycle, 3-phase generators of 1,000 K.W. each, and some 20 motors, varying in size from the 700 H.P., 150 rev. per min., which drives the 60-ft. sand-wheel, to those of 20 H.P. The 700-H.P. motor is shown in Fig. 6.

*Locomotives.*—Electric underground-traction has surpassed all other methods of power-haulage. The economy of quick transportation and the increased output that can be obtained by power-haulage are too well recognized for discussion. The cost per ton of coal hauled in 10 Pennsylvania mines was 2.7 cents by electricity, and 7.8 cents by mules.

In view of the large number of electric locomotives used in coal-mines, it is somewhat surprising that more are not employed in metal-mines. There are large, electric haulage-plants, however, in some of the more progressive mines. In the Lake Superior region, the Pittsburg and Lake Angeline Mining Co. have eight, and the Quincy Copper-Mining Co. will shortly have 15, locomotives in their underground-workings.

*Drills.*—Drilling is one of the few kinds of work in which a reciprocating motion is likely to survive. The rotary motion of the electric motor, admirably fitted to drive most kinds of machinery, is here at a disadvantage.

The air-drill has very low efficiency as an engine, but it does the work required of it, and stands a great deal of hard usage. Until the electric drill has proved itself equally durable

the air-drill is likely to hold its place; and so long as compressed air is required for drilling, it will naturally be employed for other purposes for which it is not equally suitable.

There are to-day a number of successful electric drills on the market, each of which will do the same work as an air-drill with but one-tenth the power.

Fig. 16 shows three types of electric percussion-drills,—the Box, the Gardner and the Durkee. It will be noted that in one of them (the Box) the motor is mounted directly on the drill itself, while in the other two the motors are separate from the drills, with which they are connected by flexible shafts. Each form has some advantages. The direct-connected motor avoids loss of power in the flexible shaft. A separate motor, however, divides up the weight of the outfit, and is not subjected to the vibration of the drill.

In addition to the lower cost of power, the electric drill has the advantage of avoiding the loss of efficiency which occurs in air-plants at high altitudes, and the difficulty experienced through the freezing-up of the exhaust. The cold and foggy air from drills is insufficient in quantity to have much value for ventilation; and, under any circumstances, such a method of introducing air into a mine is extravagant. A low-pressure blower driven by an electric motor, such as that illustrated in Fig. 14, will supply a large volume of air at a minute fraction of the cost required to deliver air through air-drills; and, moreover, this air will be in proper condition to breathe.

In my judgment, the abandonment of compressed air in mining is only a matter of time. What can be gained by substituting electric- for air-power was strikingly shown by the tests made by Mr. Lewis Searing of the pumping-plant of the Colorado Fuel and Iron Co.'s mines at Rouse, Colo. It was decided to replace a compressed-air system by an electric one, and, upon careful experiments made to determine the efficiency of both methods in actual operation, it was found that with compressed air 10 times the coal had to be burned under the boilers that was required with electric power, the work being the same in both cases.

The economy and convenience of electric power are so evident, and the opportunities for its profitable employment in mines so many, that the time is not distant when it will be the only system of power-distribution considered by the mining engineer.

## Application of Electricity in the Anthracite Coal-Field of Pennsylvania, with Special Reference to the Wyoming Field.

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(Albany Meeting, February, 1903.)

### INTRODUCTION.

THE term "Anthracite Coal-Field" is generally used to refer to a comparatively small territory lying in the eastern-central part of Pennsylvania. This territory includes about 3,300 sq. miles of area, located chiefly in five counties in which anthracite-mining is the dominant industry.

Although an area of 3,300 sq. miles is embraced in the region, less than one-fifth of this area, or only 484 sq. miles, is underlain by workable coal-seams. Moreover, this productive portion is not a continuous area, but consists of a number of detached basins. These are known as the Northern, containing 176 sq. miles; Eastern-Middle, containing 133 sq. miles; Southern, containing 181 sq. miles; and Western-Middle, containing 194 sq. miles. From the standpoint of business and trade, the region is divided into three districts, known as (1) the Wyoming, which includes the Northern field and the small Bernice basin; (2) the Lehigh region, which includes all of the Eastern Middle field and that part of the Southern field east of Tamaqua; and (3) the Schuylkill region, which includes the Western-Middle field and that part of the Southern field west of Tamaqua. The relative commercial importance of these districts at the present time is shown by the coal-shipments as given in the following table:

Schuylkill Region.			Lehigh Region.		Wyoming Region.		Total.
Year.	Long Tons.	Per Ct.	Long Tons.	Per Ct.	Long Tons.	Per Ct.	
1895	14,269,932	30.68	7,298,124	15.69	24,943,421	53.63	46,511,477
1900	13,502,732	29.94	6,918,627	15.33	24,686,125	54.73	45,107,484
1902	9,332,866	29.74	2,532,797	8.03	19,636,155	62.23	31,551,818

The mines are owned principally by the Delaware, Lackawanna & Western Railroad; Delaware & Hudson Canal Co.; Erie Railroad; New York, Ontario & Western Railroad; Pennsylvania Railroad; Lehigh Valley Railroad; Central Railroad of New Jersey; and the Philadelphia & Reading Railroad. The latter three are practically under one management, so that the greater part of the mining is in the hands of six large corporations which mine at least 90 per cent of the output. All of these corporations, excepting the Philadelphia & Reading Railroad, hold and operate large mines in the Wyoming district.

Not only is the Wyoming field commercially the most important of the anthracite fields, by reason of its shipping more than 50 per cent of all the coal shipped, but, considered from the standpoint of electrical equipment, it probably contains over 90 per cent of all the electrical machinery installed to date; hence, we have confined ourselves mainly to this one field. The reason for this is largely due to the conditions which have favored the introduction of electrical power in the Wyoming field; the seams are flatter here than in the other regions, and more regular, while, as a rule, less gas is present in the workings, particularly in the upper part of the field. The Delaware, Lackawanna & Western Co. has installed an extensive electrical equipment in all classes of mining-machinery applicable to anthracite mining at the present time, and this will explain the frequent reference to this one company, which probably uses more electrical power than all the other anthracite companies.

The conditions in the anthracite field have changed materially during the past 20 years, as is shown by the following figures given by H. H. Stoek, in a Bulletin on the Anthracite Coal-Field recently issued by the U. S. Geological Survey.

"In 1880 coal-royalties were from 20 to 25 cents per ton for prepared sizes, no account being taken in many cases of sizes below chestnut, and upon some of the old culm-banks a large amount of chestnut even was thrown. Royalties are now from 40 to 50 cents, and in occasional instances even higher, for prepared sizes above pea, 25 cents for pea coal and about 10 to 12½ cents for small sizes below pea. In 1877 the average number of days worked in a year was 205. In 1897 it was only 152. The cost of opening a colliery in 1887 was \$100,000; now it is from \$400,000 to \$500,000. Moreover, the coal did not then have to be so clean or so exactly-sized as now, from 12 to 15 per cent of bone being allowed in chestnut and 8 to 10 per cent in stove, as against 6 per cent and 3 per cent, respectively, now. Hence, at present about 14 per cent of the product which was for-

merly salable must be thrown upon the dump. In 1887 the average daily breaker output was 500 tons, with a maximum of 1,800 to 2,000 tons, while in 1897 the average of all breakers was 880 tons a day and the maximum 2,600 tons and over, which maximum has been increased to 3,000 tons in 1900, while the complete cost of equipment of a colliery in the latter year was probably \$700,000. In 1880 88 per cent of the coal sold was of the size which was sold at a profit, while in 1897 only 70 per cent of the total sales were of profitable selling sizes. In 1877 38 per cent of all coal mined was grate and larger sizes, which required practically no crushing or preparation for sale. By 1887 this amount had decreased to only 26 per cent; in 1897 it was 15 per cent. The selling price of prepared sizes at tide-water was \$3.26 in 1877, \$3.67 in 1887, and \$3.46 in 1897.

"The coal near the surface and in the thicker seams has in many places been worked out, necessitating deeper workings, heavier machinery and increased costs. In many ways in connection with mining, while the greater purity of the salable product has greatly increased the cost of preparation."

These changed conditions have necessitated the introduction of every possible economy in the mining and preparation of the coal. This fact, together with the concentration of ownership and the advantageous location of so many collieries in a small area, have naturally led to concentration in the mechanical equipment of the collieries.

Centralization seems to be the present policy of the large companies, and central plants are rapidly replacing smaller individual plants. A single large breaker now does the work formerly done by several smaller ones; central tubular boiler-plants have replaced half-a-dozen or more scattered plants of the old-fashioned cylindrical boilers; and large central pumping-plants or drainage-tunnels are now draining whole basins, where formerly a number of small plants were required to do the work.

One of the most complete examples of centralization is that proposed by the Delaware, Lackawanna & Western Co., in the Keyser Valley, near Scranton. This company has a number of collieries, pumping-stations, washeries, and a coal storage-plant located within a small area, and it is designed to supply them with steam- and electric-power from a central station. The boiler-plant, known as the Hampton, was put in operation during the spring of 1902, and a complete description of it will be found in *Mines and Minerals* for March, 1902.

It contains fifteen 325-H.P. Babcock & Wilcox boilers,—a total of 4,875 H.P. The fuel used is No. 3 washery buckwheat coal reclaimed from old culm-dumps. This coal is dumped

from railroad-cars into the coal-pockets, running underneath a trestle which extends the full length of the boiler-house. It feeds directly by gravity from the coal-pockets into McClave stokers. Both forced and induced draft are provided, and the gases of combustion pass through economizers. The ashes drop into a pit running underneath the boilers and are then elevated and dumped into a railroad-car running on the same track as the fuel-cars. This boiler-plant furnishes steam for the plants shown in the following table :

	Distance from Boilers. Ft.	Diameter of Pipe. In.
Sloan breaker and shaft, . . . . .	1,420	8
Hampton breaker, shaft, washery and storage, . . . . .	1,400	12
Hyde Park breaker and shaft, . . . . .	3,140	8
Central breaker and shaft, . . . . .	1,400	8
Continental hoisting and pumping shaft, . . . . .	1,500	8

It will also furnish steam for a central electric-power-station, which will be located in close proximity to the boiler-plant, and will contain three 500-K.W. Turbo alternators with switch-boards and exciters for the generating station, and there will be four sub-stations, one of which will have one 200-K.W. rotary converter with switch-board, and three of which will have three 150-K.W. rotary converters with switch-boards. The dynamo will be driven by General Electric Co.'s steam-turbines, which will be the first installation of this character in connection with coal-mining.

The application of electricity will be here taken up under the headings Mining, Haulage, Hoisting, Pumping, Signaling, Lighting, and Preparation of Coal in Breakers and Washeries; first giving consideration to the conditions which exist, favorable or unfavorable to the installation of electric appliances, and then giving typical examples of past and present equipment, and as far as possible the present equipment.

## I. MINING.

The room- and pillar-system of mining is universally used, and, as the measures have in many cases a decided pitch, the workings are laid out in lifts, and the chambers or breasts are turned off on only one side of the haulage-ways.

The method of getting coal from the face to the gangways



depends upon the inclination of the seam. For inclinations between  $10^{\circ}$  and  $18^{\circ}$ ,—that is, after mule-haulage to the face becomes impossible, and until the coal will slide in chutes,—the coal is brought from the face to the gangway in buggies hauled to the face by a hand-windlass, for which it is now proposed to substitute a windlass operated by compressed air or electricity.

The coal is usually drilled with hand or breast auger-drills and is blasted from the solid, a large amount of black powder (or occasionally dynamite) being used; and it is only recently that attempts have been made to undercut.

So far as we know, the Temple Iron Co., of Scranton, was the first to attempt the undercutting of anthracite; and the following account of its experiments has been given me by Mr. S. B. Thorne, its general manager:

“The following are the results of our tests made at Mount Lookout Colliery with undercutting-machines. We first put these machines in operation in the summer of 1901, in the Marcy seam, which is at that colliery about 4 ft. in thickness; the upper bench of coal being 1 ft. thick, directly under which is 1 ft. of bone, while 2 ft. of coal make up the bottom-bench. We used the Jeffrey Mfg. Co. machine of two kinds, one compressed-air machine (their 16-D pattern) undercutting to a depth of 5 ft. and a width of 44 in. A short time afterwards we also experimented with their 17-A electric-machine, having the same size of undercut. The last-named machine, although considerably more powerful than the former, we were forced to discard in a very short time, owing to the absolutely unbearable amount of dust which was made by it in the process of cutting. Doubtless the air-machine made as much of this dust as did the electric, but owing to the exhaust from the former the air was kept in some sort of circulation, and it was possible for the men to work in it, whereas in the case of the electric-machine it was practically a physical impossibility.

“In addition to this, our electric system at that colliery is of five hundred (500) volts, which is too high for safety where men have to handle the machine as often and with as little care as would be necessary in the case of the undercutter. There were, therefore, no tests made with this machine (the electric), although, as I say, it really did more effective work, and if a spray of water was introduced in the cutting-face to lay in some part the dust, it might still be the better of the two, if used on a two-hundred- (200) volt circuit.

“Tests made of the results as shown by one cutting across the face of a chamber 24 ft. wide with a machine, compared to the same work done under the old method and with as nearly as possible the same depth of holes, show that in the first case, out of 12 cars of material loaded out and weighed, in each case, the cars from the machine-mining averaged 2,803 lb. of coal and 158 lb. of culm, as against 2,953 lb. of coal and 52 lb. of culm by the old method. The product of the cuttings made by the machine in one trip across the face, undercutting to a depth of, approximately, 5 ft., was 2,262 lb. of material, made up of stove and larger sizes, 179 lb.; chestnut, 212 lb.; pea coal, 289 lb.; buckwheat No. 1, 353 lb.;

buckwheat No. 2, 417 lb.; and culm, 812 lb. This was made with so-called Jeffrey 'five-tooth' chain. Later on, with its 'three-tooth' chain, we secured some better results,—that is, these undercuttings contained a larger percentage of good-sized coal and less culm, but no tests were made to show exactly what the difference amounted to.

"It was very difficult to secure any correct data regarding the amount of work which could be done by one of these machines in a day. The most that we have succeeded in cutting was 75 ft. of face in 10 hours, but the average fell very far below this amount, probably not to exceed 48 ft. or two (2) chambers. This was due very largely to the sulphur, which we are considerably troubled with in the bottom-bench of this seam. It seems to come and go with no regularity. A place may advance 40 or 50 ft. without a sign of it, and then find sulphur in 5 or 6 ft., only to run out of it again immediately; and there seems to be trouble with it almost continuously. Where this sulphur was encountered, the machine could do but very, very little, and this was largely the reason for our not adopting the machines more generally. The labor situation, too, was very unsettled at the time the experiments were made. The Union was very much against the installation of the machines, and it was difficult to get any one to blast down the coal or load it after one of them. The lessened output from the car would also have been a serious question; but I feel that, with the percentage of smaller sizes decreased by the use of the three-tooth chain instead of the five-tooth, and the larger percentage of coal in the seam, which could have been saved from the gob by blasting the coal down after it had been undercut, instead of blowing it out with powder, the machines might in any ordinary seam give good satisfaction.

"I understand that several other companies in this vicinity are experimenting with the machine now, or shortly intend to, and I think it is very likely to give good results in some places, especially as the company builds them stronger to meet the conditions in the anthracite field.

"Figuring on the probable cost of mining with the machines, we came to the conclusion at the time we were experimenting with them that, charging off 25 per cent for depreciation on the machines annually, and 10 per cent on the compressor-plant, we would come out just about even with the old method; the advantage to the company being that their output would be secured with a smaller working-force,—an important consideration where men are scarce,—and that the earnings of the men would be proportionately greater."

During the summer of 1902, and just about the time the strike of that year was declared, the People's Coal Co., of Scranton, of which Mr. J. L. Crawford is president, in connection with the Jeffrey Mfg. Co., and Mr. James Pollock, mechanical engineer, experimented with a compressed-air undercutter. A change was made in the teeth of the machine, a sharp pick-tooth being substituted for the ordinary flat-tooth of the bituminous machine; and the teeth were arranged in staggered rows and inclined forward at an angle of 45° in the direction in which the chain ran. The object of this was to break the coal in lumps and not to pulverize it, as anthracite-

dust is practically a dead loss. Mr. Pollock estimates that 50 per cent of the cuttings made with this type of tooth is stove-, chestnut- and pea-coal.

A very great drawback in the undercutting of anthracite is the excessive amount of dust made. To obviate this it is proposed to inject a water-spray by means of compressed air into the undercut; but this has not yet been tried. It will thus be seen that at the present time the undercutting of anthracite, either by electric or compressed-air machines, is experimental; and while those in charge of the experiments are confident that all the difficulties will be overcome, it is yet too early to predict the outcome. Moreover, the labor-question is one which will have to be very seriously considered, and will have a material bearing upon the introduction of such machines, even if they are perfected mechanically.

Several kinds of rotary drills have been used, operated both by electricity and compressed air. These are similar to the type made by the Jeffrey Mfg. Co., and shown in Fig. 1. About 27 of these drills have been sold for use in this region; but, owing to labor-troubles, very few of them are now operated. There is no difficulty whatever in the use of such drills, and whenever the labor-situation becomes settled they will, undoubtedly, be largely used.

The Gardner Electric rock-drill (Fig. 2) has been used by the Pennsylvania Coal Co. for some months past, and the reports received from that company are very favorable, from 40 to 60 ft. being cut per day in hard sandstone. Influenced by the experience of the Pennsylvania Coal Co., several other companies have ordered these drills, and further data regarding them should be available in the near future.

## II. HAULAGE.

The conditions are very favorable for power-haulage of some sort. Owing to the irregularity of the measures, many of the gangways are extremely crooked, and therefore unsuited for rope-haulage, so that the mule is usually superseded by either electric or compressed-air haulage. Fig. 3 shows the first electric mining-locomotive used in America. It was installed by the Lykens Valley Coal Co., one of the coal operations of the Pennsylvania Railroad at the Short Mountain colliery, in 1887.

This locomotive was designed by Mr. Schlessinger, and was built by the Union Electric Co. of Philadelphia. A detailed description of the locomotive and installation will be found in the *Colliery Engineer* for September, 1887. Although a very crude affair according to our present standards, this and a similar locomotive are still in service. Another pioneer locomotive (Fig. 4) was built in 1889 by the Thomson-Houston Electric Co. for the Hillside Coal & Iron Co., and installed at the Erie colliery near Carbondale. This locomotive had a wheel gauge of only 36 in. and was specially designed for narrow gangways. It had 40-H.P. capacity and was designed to use a current at a pressure of 220 volts. It is still in service. Another early locomotive of the terrapin-back type was installed by the Hillside Co. at its No. 2 shaft at Forrest City about 1890.

In striking contrast to these early types are the later locomotives shown in Figs. 5 and 6, representing a type of General Electric locomotive installed in 1898 by the Delaware, Lackawanna & Western Co., at its Woodward mines, located at Kingston. This haulage-plant may be considered as a fair type of the electric haulage-plants of the anthracite region, and a brief description of it follows:

The power-house, which stands about 100 ft. from the No. 2 shaft, is built of brick and stone and is 25 by 45 ft. The generating-plant consists of one General Electric 8-pole, 210-H.P. 250-volt generator, directly connected with an 18 by 16 in. American-Ball engine, running at 240 revolutions per minute, and also one 330-K.W. (442-H.P.) General Electric 250-volt generator, direct-connected to a 24.5 by 22 in. McEwen single automatic engine, running at 165 rev. per min. The steam-pressure is 120 lb. per sq. in. The switch-board is of the skeleton type, and cables are carried from the generator beneath the floor. The circuit from the generator to the switch-board is provided with a circuit-breaker, which automatically cuts off all connection between the generator and the board in case conditions arise which might injure the apparatus. Each feeder running from the board to the mine is similarly equipped, thus affording ample protection to the electrical apparatus in the station. Two circuits are run from the power-house to the shaft. These have a common negative of three wires, but have separate positive feeders from the circuits which

run to the Baltimore and Red Ash seams, worked from the same shaft. The negative consists of three No. 0000 wires. The Baltimore feeder is a single wire of the same cross-section, and the Red Ash seam circuit consists of two No. 0000. These wires and the method of supporting them at the head of the shaft are shown in Fig. 7. At the head of the shaft, the negative conductors are connected to a bare stranded copper-cable of 600,000 circular mills cross-section. The positive feeder to the Baltimore seam is connected to a No. 0000 rubber-covered stranded copper-cable, and the Red Ash seam to a similar cable of 400,000 circular mills. The shaft is 1,004 ft. deep, and the wires are run in a pump-way, being supported from 12- by 12-in. oak-beams fastened across the shaft. Each cable passes over a roller made of hard wood 6 in. in diameter and 6 in. long. The Baltimore seam is 700 ft. from the top of the shaft, and the cable to it weighs 600 lb. The cable to the Red Ash weighs 1,400 lb., while the negative cable weighs 1,800 lb.,—a total of 4,000 lb. The two feeder-cables are incased in a 3-in. conduit of fiber, as the shaft is very wet. At the Baltimore seam a junction-box is placed in the fiber-conduit, and the No. 0000 cable brought out through it. At the Red Ash seam, the cables are connected in No. 0000 solid, double waterproof conductors, and carried a distance of 6,000 ft., to the end of the trolley-line. The negative conductors are connected to the track by four No. 0 bare copper-wires, secured by channel-pins to the rails; track is 40 lb. T-rail, and bonded throughout with No. 0 tin-copper wire, held in the rails with channel-pins. It is also cross-bonded at every fifth rail and at the switches. The maximum grade is about 5 per cent. A more detailed description of this plant will be found in *Mines and Minerals* for August, 1898.

The haulage-plants in the region at the end of 1902 are given in the table on page 521.

Up to the present time the mine-cars have been gathered from the rooms with mules or with a small compressed-air locomotive. (See *Mines and Minerals*, December, 1898, for detailed description of the system.) The Delaware, Lakawanna & Western Co. and the General Electric Co. have, however, for some time past, been experimenting with a cable-reel device which can be attached to an ordinary mine-locomotive, thus

*Table showing the Variation in the number of Mules and different forms of Motor-Haulage at the Anthracite mines of Pennsylvania from 1897 to 1902.*

	Mules.	Steam Loco's.	Electric Loco's.	Compressed-Air Loco's.
1897.....	15,877	*	*	*
1898.....	15,907	*	11	7
1899.....	15,690	352	29	24
1900.....	15,708	365	38	30
1901.....	16,059	362	40	51
1902.....	16,139	373	53	55

permitting the locomotive to go to the face with power transmitted to it through the cable from the line in the gangway. This device has been described by Mr. W. B. Clarke in a paper read before the present meeting.†

### III. HOISTING.

Next to haulage, probably the most extensive use of electricity is for local hoisting and pumping. The structure of the field is such that there are numerous swamps or depressions from which water must be pumped and coal hoisted. These are frequently located at a distance from the mine-opening, and, owing to the ease with which electrical power can be transmitted, it is quite widely used.

The hoists usually have a single-drum friction-clutch, double reduction-gears and a series motor. When the slopes are steep two motors are used.

A hoist installed by the Lehigh Valley Coal Co. at its Maltby colliery near Wilkes-Barre is an example of the general style of hoist used throughout the region. The motor is series-wound for 500 volts, and is capable of developing 110 H.P. for intermittent work. It is made of cast-steel, and is completely enclosed to protect it from dust and moisture. The motor and drum are mounted on the same iron frame and are connected with double reduction-gears. The drum is the standard friction cone-clutch, the type made by the Lidgerwood Mfg. Co., and has two band-brakes lined with V-shaped wooden blocks. The cone-clutch and brakes are operated by a rheo-

\* No report.

† See p. 134 of present volume.

static controller, which has a magnetic blow-out for instantly destroying the electric arc when the circuit is suddenly opened.

The drum of the hoist is 48 in. in diameter with 36-in. face, and holds about 1,400 ft. of 1-in. wire-rope. Under full load the hoist is rated at a rope-stress of 5,000 lb. at an average speed of 500 ft. per minute. The lowest level from which hoisting is done is 1,200 ft. from the hoist. The slope has an average pitch of about  $5^{\circ}$  and a maximum pitch of  $8^{\circ}$ . The hoist is placed on a platform built above the gangway, with its timbers securely fastened into the rock, and above the knuckle the empty and loaded cars pass on a parting directly under the hoist, which in this case is considered more convenient than having the hoist at one side, as is usually done. A motorman stands on an auxiliary platform back of the hoist, and is relieved from the constant vibration of the hoist while running. These hoists are not usually thus placed on a platform, but on the bottom rock.

A double-motor Flory hoist has been used very successfully for series parallel control, the motors being operated in series for slow pulling around curves and in multiple for high speeds.

#### IV. PUMPING.

The conditions favorable to local hoisting apply equally to local pumping; and a large number of all of the standard types of pumps have been equipped with electric-motors.

One claim frequently made against electric machinery is, that owing to the dirt in the mines, repairs are excessive. In one instance we have been advised of a pump operated by the Scranton Coal Co., that has run continuously for 18 months with no cost for repairs or maintenance, excepting for grease and packing. For use in local dips the small pumps run by shunt-motors give very satisfactory results, for they run at approximately the same speed, with or without load, and hence can operate even when the pump lies in the water, without injury to the pump. These pumps are generally of the plunger-type, outside-packed and with shunt-wound motors. There is nothing peculiar about these installations, the pump being usually small, 4 by 4 or 6 by 6, although recently some much larger installations have been made.

Fig. 8 shows probably the largest electric-pump yet in-

stalled in the anthracite region. It is a double-acting triplex electrically-driven mine-pump with outside connected plunger, and was built by The Jeanesville Iron Works Co. of Jeanesville, Penna., for the Delaware, Lackawanna & Western Railroad Co., for the Avondale mines.

As shown in the illustration, there are six pump-cylinders placed in pairs at opposite sides of the suction-valve chambers. The rams are connected up to a three-throw crank, driven through reduction-gearing by electromotors placed as shown. The pump-plungers are each 8 in. in diameter, and have a stroke of 18 in. The crank-shaft, which is designed to run at 34 revolutions per minute, is 9 in. in diameter, and the motor-shaft  $5\frac{1}{2}$  in. in diameter. The spur-wheels on the former measure 8 ft. 10 in. in diameter by 10 in. width of face, and have 106 teeth of No. 1 diametrical pitch. The pinions on the motor-shaft are 18 in. in diameter. There are two General Electric Co.'s motors at each end of the motor-shaft, designed to give 113 H.P. each, when supplied with current at 250 volts, and built to run at 225 revolutions per minute. The pumps are intended to deliver 800 gallons per minute through 3,500 ft. of 12-in. column-pipe when working against a head of 750 ft. The suction- and discharge-pipes are each 12 in. in diameter. The water-end is wood-lined throughout for acid water. The total space occupied by the pump measures 29 ft. in length by 17 ft. in width and 9 ft. in height. The total weight of the pump and motors is 143,744 lb.

The importance of the water-problem in connection with anthracite mining is well shown by the fact that there are a number of pumping-stations having capacities of from 5,000,000 to 10,000,000 gallons per day. The Gilberton water-shaft of the Philadelphia & Reading Co. is about 1,000 ft. deep and is equipped with hoisting-buckets. The capacity of the plant is 7,000,000 gallons per day, and it is designed to drain the entire basin operated from the Draper and Gilberton collieries.

The Delaware, Lackawanna & Western Co. is now sinking a water-shaft in the Keyser Valley, near Scranton, which will probably be equipped with tanks hoisted by electric machinery. Experiments are in progress to make this hoist automatic, so that an attendant may be done away with, and the starting and stopping of the hoist regulated by means of the level of water



in the sump. This pumping-plant will receive power from the large central station before referred to, and will be another unit in the centralized equipment for the entire Keyser Valley.

## V. SIGNALING.

The electric signaling-devices in use range from simple Bell circuit-telephones to a complete annunciator signal-system. Many of the large companies now have their own telephone exchanges, and a complete system connects all of their various collieries with the main offices. In addition to these general systems, which are connected to the ordinary long-distance telephone-system, there is frequently a local telephone-system at each colliery, connecting all of the principal buildings, while many of the shafts and slopes have independent systems for communicating between the engine-room and each level from which hoisting takes place.

The Delaware, Lackawanna & Western Co., after experimenting quite extensively with various styles of batteries, telephones, etc., has adopted a high-grade, long-distance bridging telephone, wound for about 1,600 ohms resistance. Series telephones were tried, but were found unsatisfactory, owing to poor construction and because a single loose connection in one of the telephones affects all the others on the circuit. The bells found to be most satisfactory for general use are 6-in. skeleton-frame, placed with pivoted armatures wound for different resistance, according to the work to which they have to be applied. Two kinds of batteries have been selected, one for circuits that are well insulated, and the other for lines that are poorly insulated, from water or from grounds on the line. For the well-insulated circuits the ordinary open-circuit carbon cylinder-battery is used, because it is strong, is easily and cheaply recharged, and consists of zinc-cylinder cells that are short-circuited, easily cleaned and renewed. For the leaky circuits the Gordon battery is favored, as it does not polarize when the line is badly grounded, and is less expensive to operate than the open-circuit dry batteries. On a very leaky circuit the Gordon batteries have worked for about two years without recharging, while the carbon batteries formerly had to be recharged every ten days.

FIG. 1.

FIG. 2.

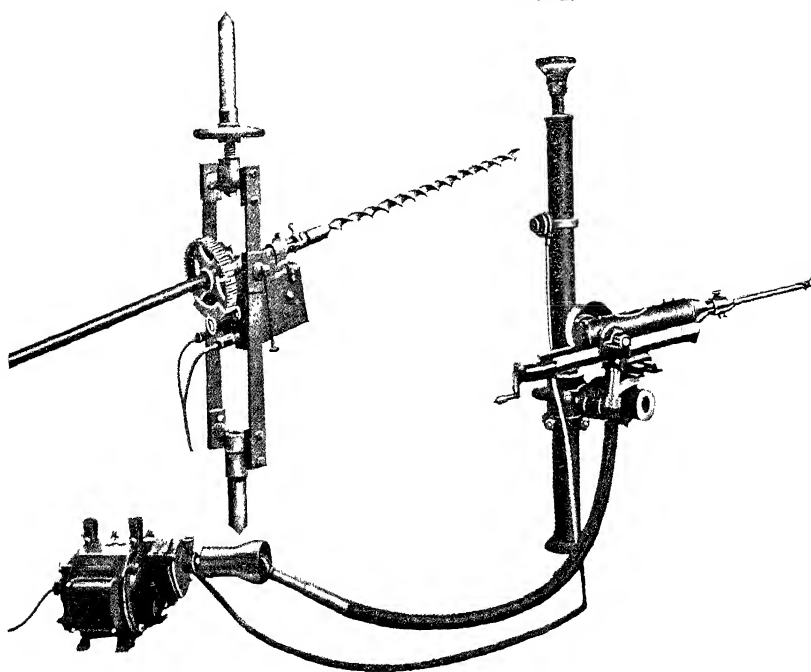
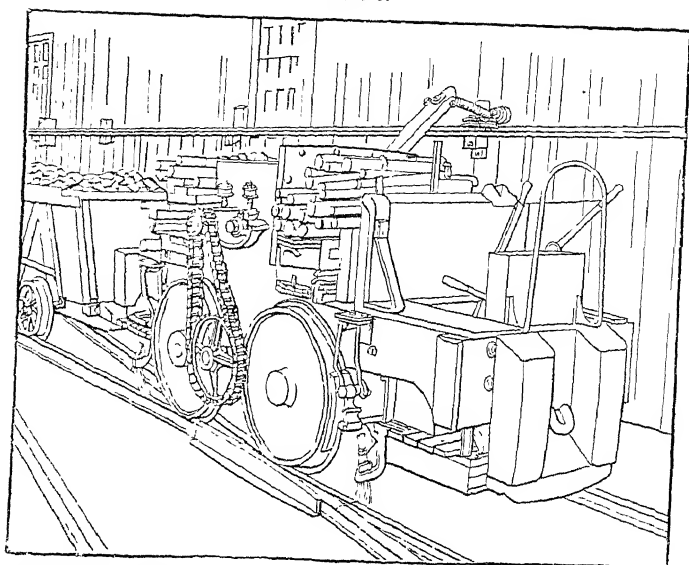


FIG. 1.—THE JEFFREY DRILL.

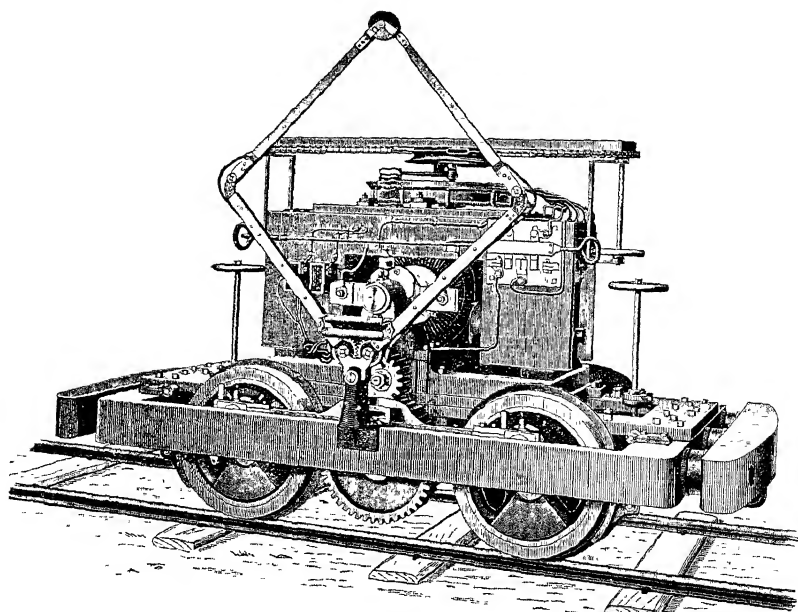
FIG. 2.—THE GARDNER DRILL.

FIG. 3.



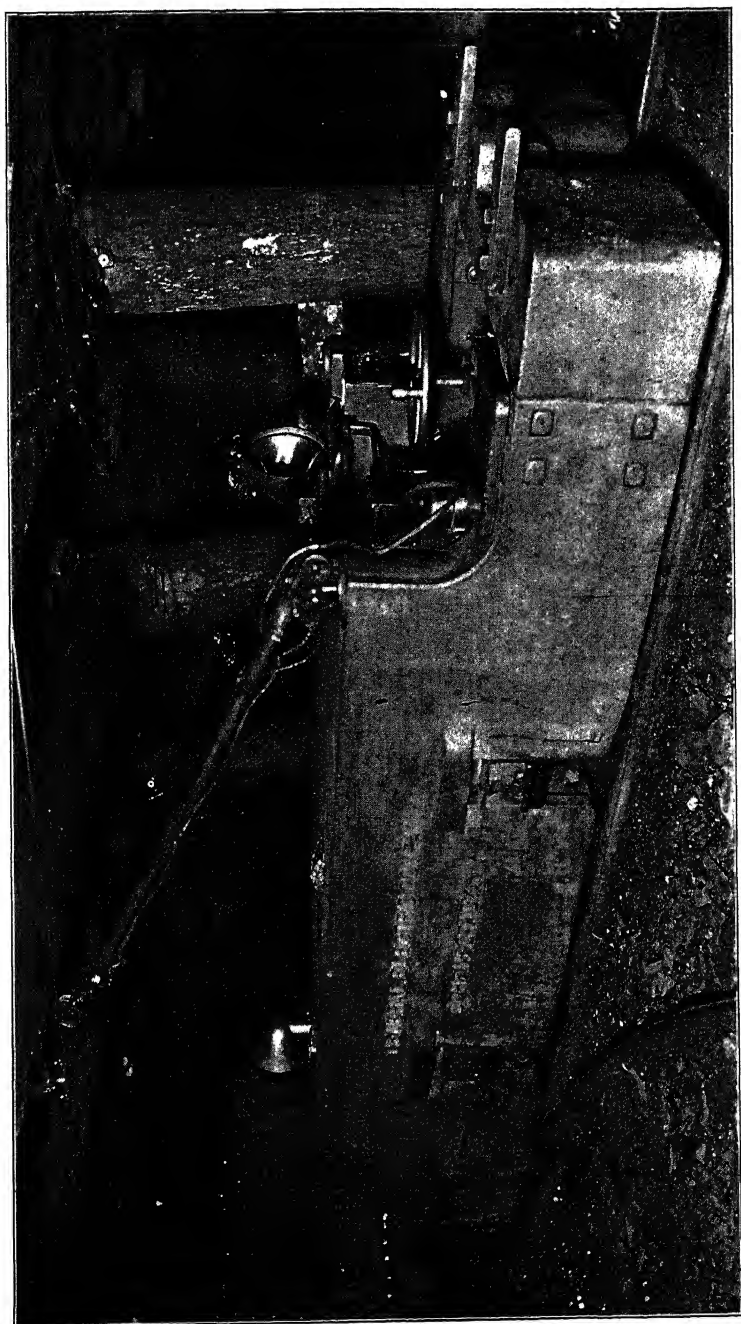
FIRST ELECTRIC MINING-LOCOMOTIVE USED IN AMERICA, 1887. (LYKENS VALLEY COAL CO.)

FIG. 4.



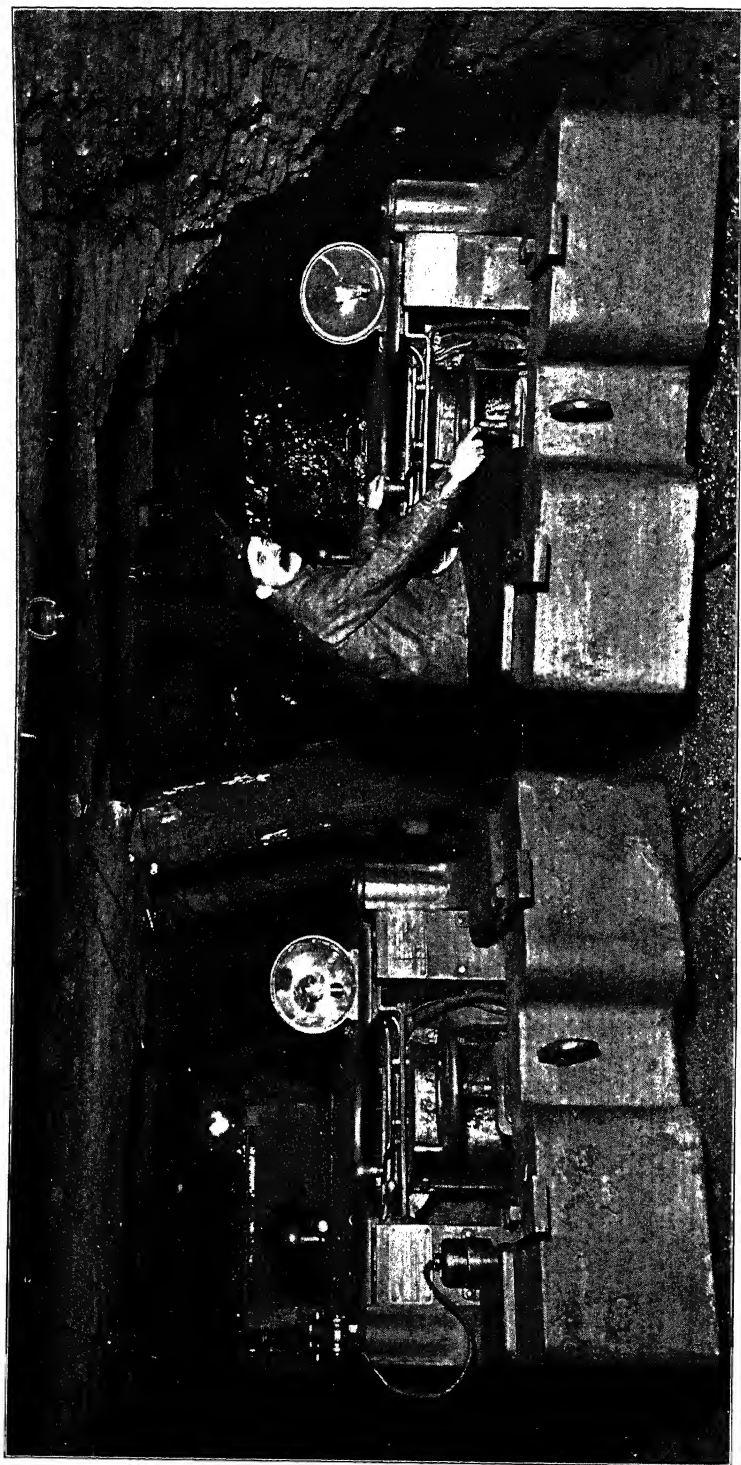
ONE OF THE EARLY ELECTRIC MINING-LOCOMOTIVES, 1889. (HILLSIDE COAL & IRON CO.)

FIG. 5.



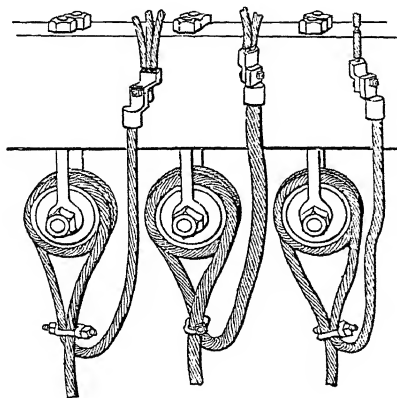
MODERN TYPE OF ELECTRIC-LOCOMOTIVE, 1898. (DELAWARE, LACKAWANNA & WESTERN RAILROAD CO., AT WOODWARD MINES.)

FIG. 6.



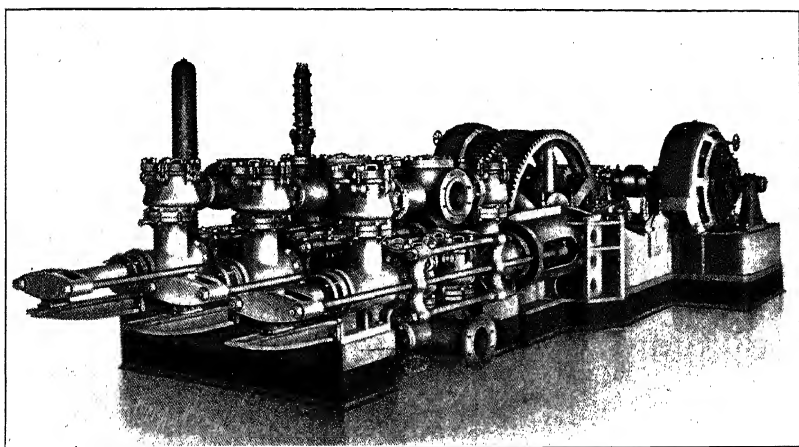
MODERN TYPE OF ELECTRIC-LOCOMOTIVE, 1898. (DELAWARE, LACKAWANNA & WESTERN RAILROAD CO., AT WOODWARD MINES.)

FIG. 7.



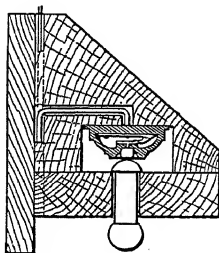
METHOD OF SUPPORTING WIRES AT HEAD OF SHAFT.

FIG. 8.



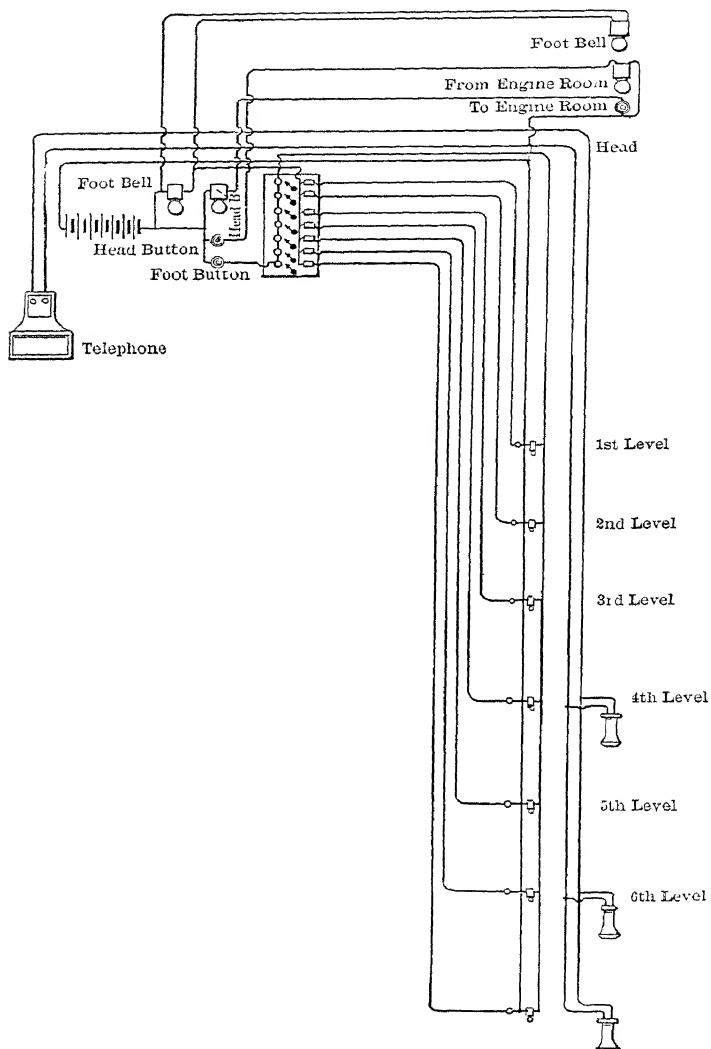
DOUBLE-ACTING, TRIPLEX, ELECTRIC-DRIVEN MINE-PUMP. (DELAWARE, LACKAWANNA &amp; WESTERN RAILROAD CO., AT AVONDALE MINES.)

FIG. 9.



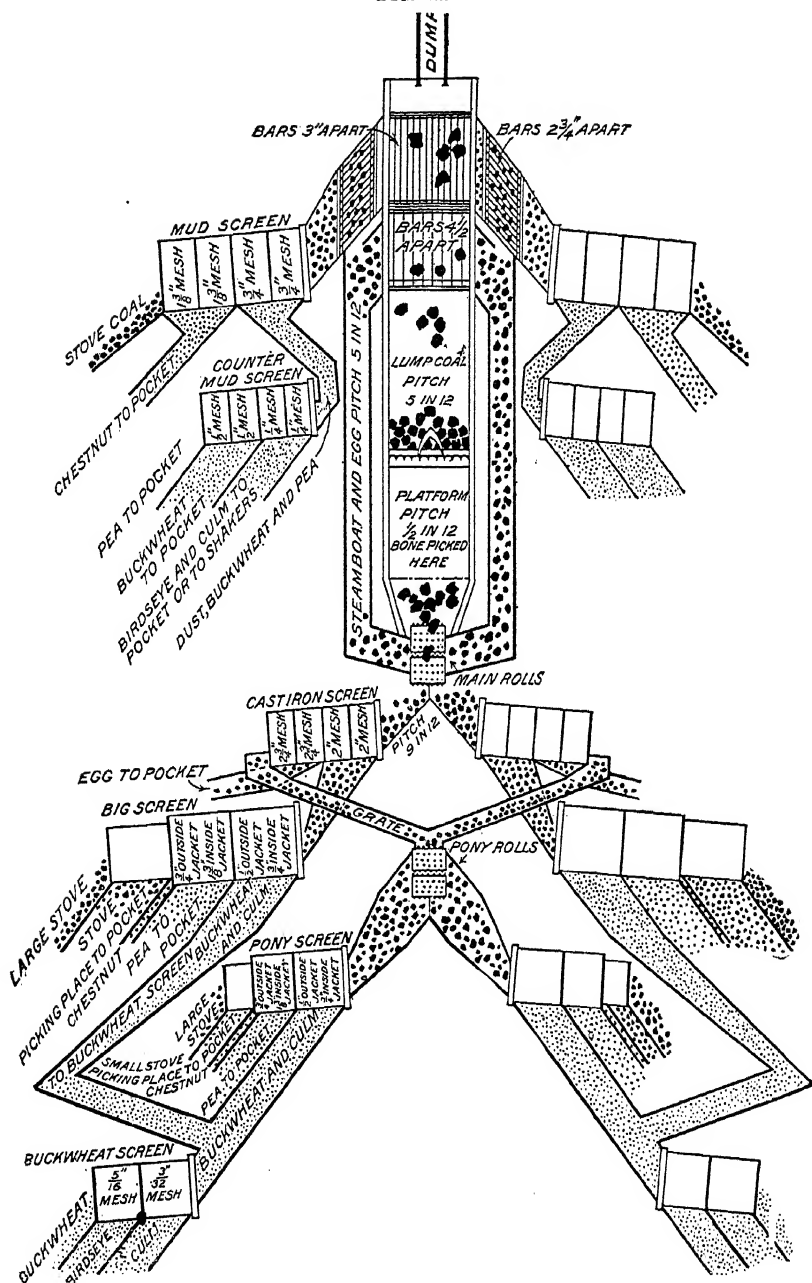
ELECTRIC PUSH-BUTTON FOR USE IN MINES.

FIG. 10.



ELECTRIC SIGNAL SYSTEM AT THE AUCHINCLOSS AND BLISS MINES

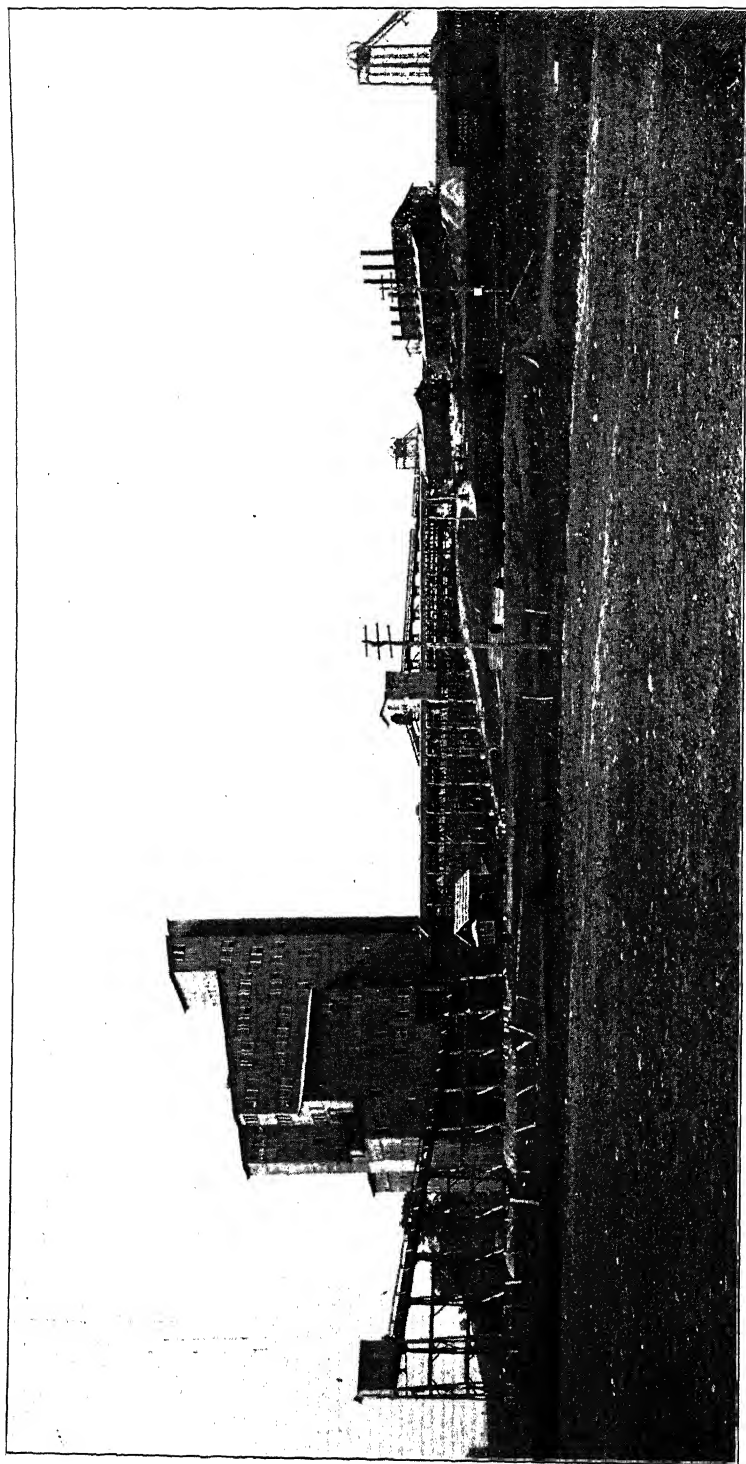
FIG. 11.



SKETCH OF GENERAL METHOD OF PREPARING ANTHRACITE IN THE BREAKER.



FIG. 12.



GENERAL VIEW OF THE PLANT AT AUCHINCLOSS COLLIERY.

A very durable and cheap push-button made at the mines (Fig. 9) consists of a base made out of an old 3-in. rubber pump-valve, with brass castings fastened on opposite ends of the same face. Attached to one of these is a strip of spring-brass  $\frac{3}{4}$  in. wide and  $\frac{3}{8}$  in. thick. This strip is made to pass over the other casting with a contact-face for closing the circuit, as the circuit-wires are attached to the castings on the leather. These have been found to be the most durable of all the buttons tried, and are cheaply made.

For slope-signaling and similar work, bare galvanized wire has been used. The wires are stretched about 6 or 8 in. apart, and when the runner wishes to signal to the engineer he simply presses the two wires together the proper number of times, according to where he is standing.

The most extensive system of signals in use by the above company is at the Auchincloss and Bliss mines, where the two shafts and engine-rooms are equipped with a complete system of call-and-reply signals between head and foot of shaft, these being supplemented with telephone connections.

At this mine there are two shafts; No. 1 is 1,720 ft. deep and No. 2, 1,690 ft. deep, the first seam being about 700 ft. below the surface in No. 1 and 630 ft. in No. 2. Seven seams are operated from each shaft, and the signals for the several landings are arranged in each shaft, as shown in Fig. 10.

Each landing has a separate call to the engine-room, and rings a bell at the same time at the head of the shaft. There is a separate call-and-reply system between the engineer and the head-tender at the shaft.

There is a telephone in each engine-room and one at each landing, and telephone-connection between the two shafts on the surface by an extra cable from No. 2 engine-room to No. 1 shaft. There is also telephone-connection from No. 1 engine-room to the mine office, about three-quarters of a mile away. A complete description of this installation will be found in *Mines and Minerals* for March, 1899.

At the Ontario colliery of the Scranton Coal Co., Mr. Geo. Hawley, electrical engineer for the company, has installed a very satisfactory system of signaling by means of lamps. Four lamps are placed in one circuit in a series and are located as follows: One at the foot of the shaft, one at the head, one in

the headman's shanty at the top, and one in the engine-house. These are worked by a switch located at the foot of the shaft, and when the cage is ready to be hoisted the footman throws on the circuit by means of this switch. The lamp at the foot shows that the circuit is working, while the lamps at the head and in the headman's shanty and engine-house show the headman and the engineer that the footman is ready to hoist. Before hoisting, however, the engineer must receive a bell- or hammer-signal from the headman.

## VI. LIGHTING.

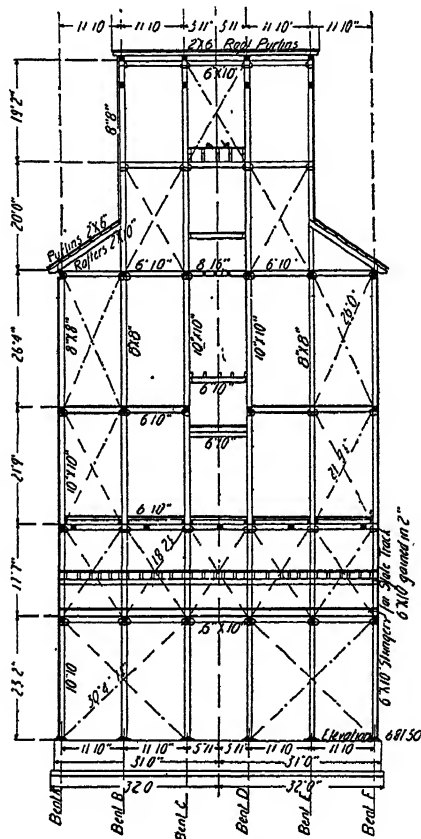
Electric lighting is very general about the breakers and all of the surface-buildings; in many cases it extends to the bottom of the shaft or slope, and in some instances along the main gangways. Most of the mines are lighted from city lighting-stations or from their own plants, generally the latter.

## VII. PREPARATION OF COAL.

Until recently the only application of electricity about the breakers was an occasional electric-hoist and electric-lights, but this is a field to which the applications of electricity may possibly be largely extended in the near future, particularly if an extensive experiment now being made by the Delaware, Lackawanna & Western Co. is successful. In order to appreciate the opportunities for electric installations, let us briefly consider the method of preparing anthracite in the breaker, as shown in the accompanying diagram (Fig. 11). The coal is hoisted to the top of the breaker either directly from the shaft or by means of a separate breaker-hoist, or by conveyor-lines. At the top it is dumped upon bars, which in some cases are oscillating. It then successively passes through crushing-rolls which break it, and then over shaking-screens or through revolving-screens which size it. Before reaching the shipping-pockets it must be picked, and in many cases this is done by machinery, while the smaller sizes are jigged. It will thus be seen that there are a large number of separate pieces of machinery in a breaker. Heretofore, these have been run by gearing, by belting, or by rope-drives. The necessity of providing space for any of these forms of transmission is quite a serious problem in the design of a breaker. It is evident,

therefore, that any system by which power can be applied directly to each individual piece of machinery and as easily carried as in the case of electric-wires, offers many advantages. In the spring of 1902 the Delaware, Lackawanna & Western Co. put into operation at the Auchincloss colliery an experimental breaker (Figs. 12, 13 and 14), which has a number of

FIG. 13.



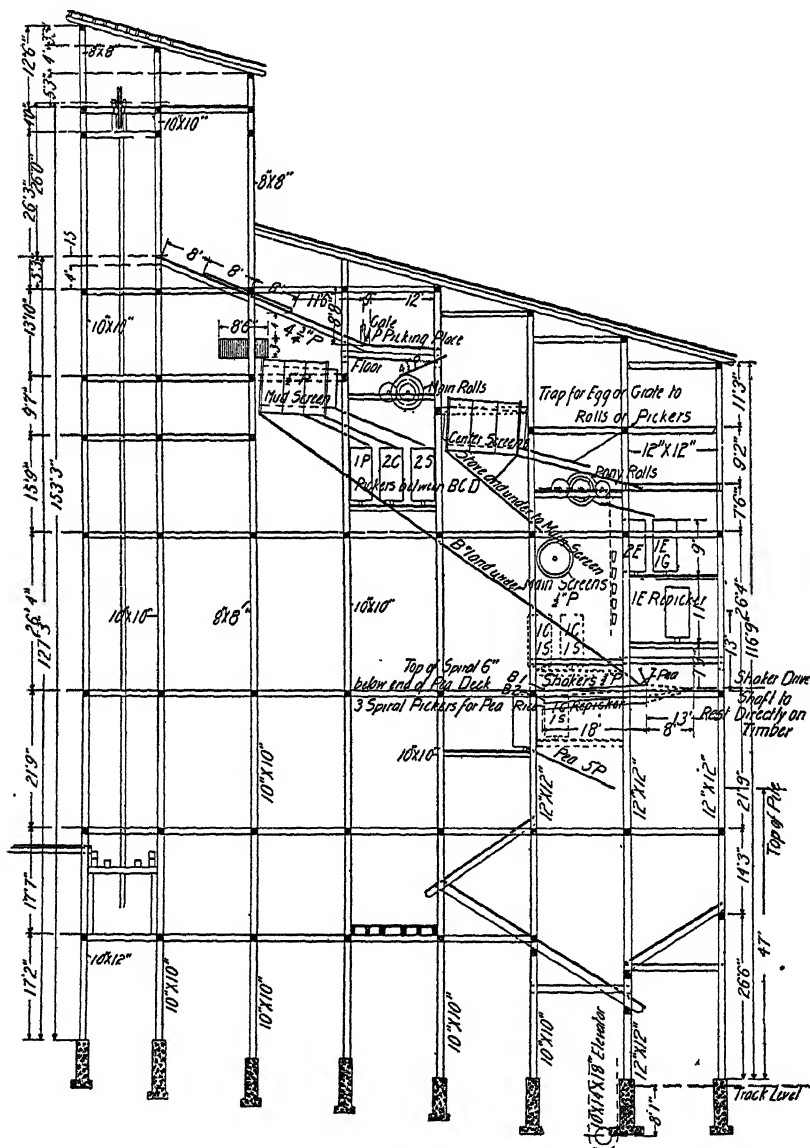
**End Elevation.**

### EXPERIMENTAL BREAKER AT AUCHINCLOSS COLLIERY.

unique features in its construction, but which is principally characterized by the fact that it is electrically operated throughout, each piece of machinery being operated by an independent motor. Fig. 12 is a general view showing the arrangement of the plant. The sections (Figs. 13 and 14) illustrate strikingly the contrast between an electrically-

equipped breaker and one equipped with gearing and belting, the absence of the vibration caused by shafting and belting

FIG. 14.



permitting the use of much lighter framing in the electric-breaker.

The electrical equipment of this breaker is as follows :

	K.W.
1 M.P. 4-45-975 generator, belted to Ball & Wood engine, .	45
1 M.P. 6-50-280 generator, direct-connected to 13 x 12 Payne engine, . . . . .	50
	<hr/> 95

### *Motors.*

	H.P.
1 M.P. 4-15-800, driving main rolls, . . . . .	15
1 C.E. 4-10-650, driving pony rolls, . . . . .	10
1 C.E. 4-10-650, driving pony rolls, . . . . .	10
1 C.E. 4-7½-815, driving mud-screens, . . . . .	7½
1 C.E. 4-7½-815, driving main screens, . . . . .	7½
1 C.E. 4-7½-815, driving center screens, . . . . .	7½
1 M.P. 4-15-800, driving shaking screens, . . . . .	15
1 M.P. 4-15-800, driving elevator-line, . . . . .	15
1 M.P. 4-25-650, driving fuel conveyor-line, . . . . .	25
1 M.P. 4-25-560, driving scraper-line, . . . . .	25
1 C.E. 4-7½-815, driving dust-pan, . . . . .	7½
1 C.E. 4-7½-815, driving empty car-hoist, . . . . .	7½
1 C.E. 4-7½-815, driving blower, . . . . .	7½
1 G.E. 53 R Y. motor, operating loaded car-hoist, . . . . .	50
	<hr/> 210

All motors are completely enclosed in No. 26 galvanized-iron cases, with a large well-fitting door and with space inside for the rheostat and switches. The cases are fairly air-tight, but are so arranged that they can be easily removed from about the motors when required. Clean air from the outside is forced into these cases from a blowing-fan under a pressure of 2 oz. This prevents the fine dust from the breaker entering the cases and interfering with the working of the motors. It also serves to keep the motors cool. Since the strike came on just as this breaker was ready to start, and the work done after resumption has not given sufficient coal to test its capacity, this device has not yet been thoroughly tried, and may still be said to be in an experimental stage.

## VIII. ELECTRICITY IN THE ANTHRACITE-MINES.

The electrical equipment of the anthracite-field at the present time is, approximately, as given in the following table :

*Present Electric Equipment in the Anthracite Coal-Field.*

	Generators.	Loco's.	Hoists.	Pumps.	Drills.
Wyoming region :					
Penn Coal Co. :					
Forest City.....	5	5	1	7	2
Erie Colliery.....	2	1			
Glenwood.....	1				
Consol. ....	1				
Scranton Coal Co. :					
Mt. Pleasant.....	1	2		1	2
Ontario.....	2	2		2	
Raymond.....	1	2		1	
Johnson.....	2	2	3	2	
Pine Brook.....	1				
D. L. & W. :					
Woodward mines.....	2	3	8	6	
Bliss mines.....	2	3	2	1	
Pettibone mines.....	2		1		
Cayuga mines.....	2	4	1		1
Avondale mines.....	1	1	2	4	
Bellevue mines.....	2	5	3	2	
D. & H.—Olyphant.....					2
Lehigh Valley :					
Maltby.....	1	1	1	2	
Twin Shaft.....	1	1		2	
Kingston Coal Co.....	1	2			
Temple Iron Co.—Mt. Lookout.....	3	3	3	1	
Green Ridge Coal Co.....	1	2	1		1
Total Wyoming field.....	34	39	26	31	9
Schuylkill region :					
Lehigh Valley Coal Co.—Centralia.....	1	2			
Enterprise Coal Co.....	2	3		1	
Pine Hill Coal Co.....	1	2			
St. Clair Coal Co.....	2	3	2		
Lykens Valley Coal Co.....	2	2			
Total.....	42	51	28	32	9

This table shows that the electrical equipment of the anthracite region is almost entirely in the Wyoming field, and in that field the Delaware, Lackawanna & Western Railroad Co. has, by far, larger installations than any other single company. A detailed statement of the equipment at the several collieries of that company will, therefore, be interesting, and is as follows :

*Woodward Mines.*

1 M.P. 10-330-165 G.E. generator, direct-connected to 24½ by	K.W.
22 McEwen engine, . . . . .	330
1 M.P. 8-165-240 G.E. generator, direct-connected to American-Ball engine, . . . . .	165
	495

	H.P.
Electric-Hoist, G.E. 51 motor, . . . . .	80
“ “ 2 G.E. 57 motors, . . . . .	128
“ “ 2 G.E. 57 motors, . . . . .	128
“ “ G.E. 53 motor, . . . . .	50
“ “ Westinghouse 39 motor, . . . . .	75
“ “ 2 G.E. 53 motors, . . . . .	100
“ “ Westinghouse 38 B. motor, . . . . .	50
“ “ 2 G.E. 53 motors, . . . . .	100
1 Electric-Locomotive, 13 tons, 2 G.E. 53 motors, . . . . .	100
1 “ “ 13 tons, 2 G.E. 53 motors, . . . . .	100
1 “ “ 13 tons, 2 G.E. 53 motors, . . . . .	100
1 Electric-Pump, C.E. 4-7½-815 motor, . . . . .	7½
1 “ “ Westinghouse 5-H.P. motor, . . . . .	5
1 “ “ T.H. 3-H.P. motor, . . . . .	3
1 “ “ Westinghouse motor, 10-H.P., . . . . .	10
1 “ “ G.E. 17½-H.P. motor, . . . . .	17½
1 “ “ G.E. 50-H.P. motor, . . . . .	50

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1,104
*Bliss Mines.*

	K.W.
1-150 K.W. Bullock generator, 225 R.P.M. direct-connected to 18 by 16 McEwen engine, . . . . .	150
1-150 K.W. Westinghouse generator, 200 R.P.M. direct-connected to 18 by 19 Ames engine, . . . . .	150
	<hr/> 300

	H.P.
1 Electric-Hoist, 2 G.E. 51 motors, . . . . .	160
1 “ “ G.E. 2000 motor, . . . . .	140
1 Electric-Locomotive, 10 tons, 2 G.E. 61 motors, . . . . .	100
1 “ “ 13 tons, 2 G.E. 61 motors, . . . . .	100
1 “ “ 13 tons, 2 G.E. 61 motors, . . . . .	100
1 Electric-Pump, Westinghouse 5-H.P. motor, . . . . .	5

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605
*Pettibone Mines.*

	K.W.
1-25 K.W. generator, . . . . .	37½
1-12½ K.W. generator, belted to 12 by 11 Ames engine, }	

	H.P.
1 Electric-Hoist, Westinghouse 39 Ry. motor, . . . . .	75

*Cayuga Mines.*

	K.W.
Generator direct-connected to M.P. 8-175-225, engine 16 by 18, . . . . .	175
1 generator, 150 K.W. belted to McEwen engine, 16 by 18, . . . . .	150
	<hr/> 325



	H.P.
1 Locomotive with 2 G.E. 53 motors, . . . . .	100
1 " " 2 G.E. 53 motors, . . . . .	100
1 " " 2 G.E. 53 motors, . . . . .	100
1 " " 2 G.E. 60 motors, . . . . .	54
1 Hoist with 1 G.E. 51 motor, . . . . .	80
	<hr/> 434

*Auchincloss Mines.*

A table of the data pertaining to the electrical equipment of the experimental breaker at the Auchincloss Mines is given on page 537.

*Avondale Mines.*

	K.W.
1-250 H.P. Crocker-Wheeler generator, 200 R.P.M., direct-connected to 20 by 20 McEwen engine, . . . . .	200
	H.P.
1 G.E. 7½-ton electric-locomotive, . . . . .	54
1 Electric-Pump, 226 H.P., 2—113-H.P. motors, . . . . .	226
1 Electric-Hoist, G.E. 51 Ry. motor, . . . . .	80
1 " " Westinghouse 38 B. motor, . . . . .	50
1 Electric-Pump, 20-H.P. Crocker-Wheeler motor, . . . . .	20
1 " " 17½-H.P. G.E. motor, . . . . .	17½
1 Electric-Pump, Westinghouse 5-H.P. motor, . . . . .	5
	<hr/> 452½

*Bellevue Mines.*

	K.W.
Generator direct-connected M.P. 10-330-165, to 22 by 24½ engine, . . . . .	330
1 M.P. 4-100-650, belted to 16 by 18 McEwen engine, . . . . .	100
	<hr/> 430
	H.P.
1 Hoist with G.E. 51 motor, . . . . .	80
1 Hoist with 2 G.E. 53 motors, . . . . .	100
1 Locomotive with 2 G.E. 53 motors, . . . . .	100
1 " " 2 G.E. 53 motors, . . . . .	100
1 " " 2 G.E. 53 motors, . . . . .	100
1 " " 2 G.E. 53 motors, . . . . .	100
1 Hoist with G.E. 57 motor, . . . . .	64
1 Pump with 1 M.P. 4-45-925, . . . . .	50
1 " " 1 M.P. 4-30, . . . . .	30
1 Locomotive with 2 G.E. 53 motors, . . . . .	100
	<hr/> 824

So far as we know no one is now contemplating the carrying of high-voltage currents into the mines and there transforming them, as has been done in the bituminous-coal regions.

The statements contained in this paper show that a large amount of experimenting with electricity is now under way in the anthracite-region, and if only a part of this is successful, the visitor to the anthracite field five years from now will see a vast change in its electrical equipment.

In conclusion, we wish to acknowledge assistance received in compiling this paper from Mr. F. J. Platt, electrical engineer, of Scranton, who has been intimately connected with many of the installations mentioned; Mr. H. M. Warren, electrical engineer of the Delaware, Lackawanna & Western Railroad (Coal Department); Mr. G. W. Hawley, electrical engineer of the Scranton Coal Co.; Mr. S. B. Thorne, general manager of the Temple Iron Co.; and Capt. W. A. May, general manager, and Mr. V. L. Peterson, superintendent of the Hillside Coal & Iron Co. Thanks are due also to the several manufacturers whose machinery is illustrated, and who have furnished the photographs or drawings from which the engravings have been made.

## The Determination of Power for Rolling Iron and Steel.

BY LOUIS KATONA, RESICZA, HUNGARY.

(Albany Meeting, February, 1903.)

THE practical execution of the design for rolling-mills presented by the writer at the Paris meeting\* of the Iron and Steel Institute in 1900 involves the determination of some questions not yet discussed, among which is the determination of the amount of power required to perform the rolling of pieces of iron and steel of given dimensions. It is necessary to know this factor if we would put motors of proper capacity to the respective pairs of rolls, or, reciprocally, if, having standard motors at our disposal, we would design grooves to suit them. Moreover, this factor is involved in giving right dimensions to every part of the rolls and housings.

Expert designers of engines for driving rolling-mills of the present type receive very indefinite directions, such as: that the motor is to drive a light, medium or heavy train; that it must work with fly-wheel or reversing gear; that the number of revolutions per minute should not exceed 40 or 60, and that at a given speed the power developed should be from 100 to 200 H. P., or from 1,000 to 2,500 H. P. It frequently happens, however, that an engine of 200 H. P. is forced under the circumstances to press or develop as much as 300 H. P., while, on the other hand, the 1,500-H. P. engine may be rolling-stock which could be worked with a maximum expenditure of 60 to 80 H. P. Of course such variations as these do not permit the best adjustment of parts or the best economy of power and fuel. The design of roll-trains is even more vaguely limited. Every part of a train, from the point where it connects with the engine to the last groove in the rolls, bears evidence that there are no rules for calculating even approximately the dimensions of the main-shafts, couplings, pinions, gears, rolls, trunnions,

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\* The paper here referred to will be found on page 259 of the *Journal of the Iron and Steel Institute*, No. 2, for 1900.

bearings, stands, etc. General usage is almost always followed; a few cases are known; and, apart from these, audacious experiments are made in altering shapes, dimensions, etc.,—a course which is not always progressive.

In my opinion an intelligent improvement is impossible without a knowledge of the power required for the work of rolling and the pressure developed on the bearings; and isolated alterations of shape or size in single parts are either not reasonable or cannot be properly carried out through the entire apparatus and within the limits of present construction. For example, it is well-known that economy of power requires the proper relation between the diameter of the rolls and the dimensions of the pieces to be rolled. But, although the diameters of the rolls now in use fail in many cases to conform to this principle, the dimensions and forms of the rolls and stands prevent any increase of diameter, however desirable. The cogging of the roll surface, employed as a partial remedy, increases, of course, the waste of power. As an example, I may mention the necks (Fig. 1) of rolls, for the calculation of which we have no definite rule. They are usually determined, therefore, only by custom, or by the conditions of casting which make the founder desire that the roll and the neck shall not differ too greatly in diameter; or by the general desire to make the neck heavy enough to resist powerful shocks from the couplings, etc. In other words, all parts of the train are empirically proportioned. •

The present paper is intended to throw some light on these questions, and to emphasize the importance of certain doubtful or unknown factors which it is hoped that other investigators will study further.

Rolling-mill practice plainly indicates that the requisite power for rolling any piece of iron is a function of the mass to be rolled, the reduction of the section, the temperature, and also, in all probability, the analysis of the material and the shape of the section. For the purpose of expressing methodically the law of these functions, we will take as a basis the simplest case (see Fig. 2), namely, that of a piece *A* of rectangular section to be rolled from a thickness *a* to *b*. We then take the instant

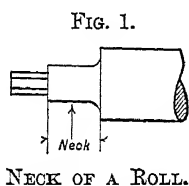


FIG. 2.

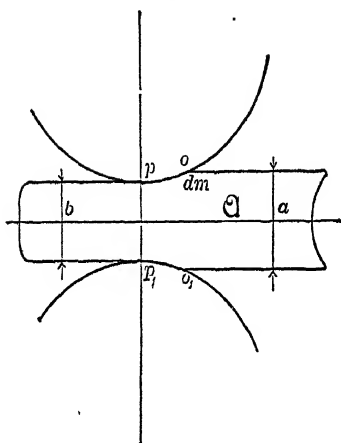


FIG. 3.

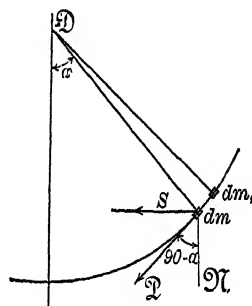


FIG. 4.

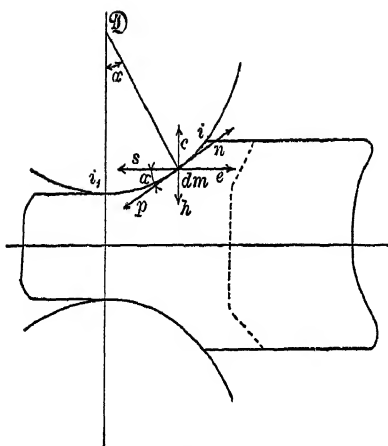
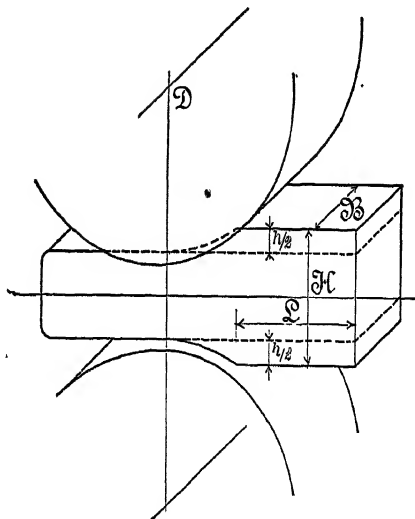


FIG. 5.



DIAGRAMS OF SIMPLE ROLLING FOR DECREASE OF THICKNESS, AND THE FORCES THEREIN INVOLVED.

when the point  $o$  of the roll comes in contact with the molecule  $dm$  of the red-hot slab, and follow this point along its course as far as contact continues in working, *i.e.*, to the point  $p$ . The work done during this period, with the assistance of the contact, includes the decrease of the thickness of the slab, and also its movement from the front towards the rear side of the train, and simultaneously the removing of the rolled portion from between the rolls and the bringing successively of the new unworked material between them. This work is done by revolving surfaces on the arcs  $op$  above and  $o_1 p_1$  below, with the aid of friction. The molecule  $dm$ , coming in contact with the roll, is forced out of its original place by the power  $P$  acting on the periphery. The measure, direction and time of displacement will depend directly upon the measure, direction and efficiency of the power  $P$ . The force  $P$  (see Fig. 3), acting in the direction of the tangent, may be resolved into a horizontal component  $S$  and a vertical component  $N$ , and the angle  $\alpha$  at  $D$ , at the center of the roll, between the vertical and radius drawn to  $dm$ , will be the same as the angle between  $P$  and its horizontal component  $S$ .  $S$ , the horizontal component of  $P$ , acting with the assistance of friction, is pulling the slab forward between the rolls; the other,  $N$ , is compressing it. The peripheral velocity being constant, it is evident that the component  $S$  proportionally increases as the slab moves forward to the point  $p$  (Fig. 2), at which point it attains the maximum, the tangent becoming horizontal. Consequently,  $S$  continually tends to tear each molecule subjected to contact from the next following one in the slab. This effect is counteracted by the tensile strength  $n$  of the material.\*

The molecular force  $n$  (Fig. 4), which counteracts the force  $p$ , may also be resolved into a horizontal component  $e$  and a vertical component  $c$ , one of which,  $e$ , resists the forward movement of the mass, while the other,  $c$ , resists its compression. The excess of  $s$  over  $e$  is the resultant that does the work; the greater this excess the more rapid the progress of rolling and the larger the number of molecules displaced in a

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\* The stress between the two contiguous molecules must be less than the tensile strength at the temperature given or else the material would be torn apart, and it must be greater than the elastic limit, because it brings about permanent deformation.

given period. All these components acting at once upon the same point, the effect of  $s$  and  $c$  may be summed up in a resultant which may be called the differential of the pressure on the bearings. The direction of this resultant force we can assume is known, for it must go through the center of the roll; its measure remains to be determined. The components  $h$  and  $e$  (Fig. 4) may also be united in one resultant, the effect of which is to change the relative position of the molecules of the slab. The following formulas express the above statement:

$$\begin{aligned}s &= p \cos \alpha. \\ c &= n \cos (90 - \alpha) = n \sin \alpha. \\ e &= n \cos \alpha. \\ h &= p \cos (90 - \alpha) = p \sin \alpha.\end{aligned}$$

Accordingly, the resultant representing the pressure on the bearings will be:

$$C_s = \sqrt{p^2 \cos^2 \alpha + n^2 \sin^2 \alpha}; \quad (1)$$

and the force performing the rolling, with respect to each molecule in the layer attacked, will be:

$$H = \sqrt{p^2 \sin^2 \alpha + n^2 \cos^2 \alpha}. \quad (2)$$

Pursuing further the course of the rolling, we see that the molecule of iron, under that piece of the arc corresponding to the angle  $\alpha$ , is set free, at the point  $i$ , from the forces communicated by contact. On its way through this arc the molecule will require at every successive position the power  $H$  to perform the rolling, and will consequently cause a certain pressure  $C_s$  on the bearings. The alteration of the angle  $\alpha$  at successive points will increase and decrease, respectively, the pressure on the bearings due to that molecule, and the force required for the rolling or forward movement; and the values of these forces at any given point are to be found by differentiating the equations 1 and 2 above on the basis of the variation of independent functions:

$$\begin{aligned}dC_s &= \frac{[2n^2 \sin \alpha \cos \alpha - 2p^2 \cos \alpha \sin \alpha] d\alpha}{2\sqrt{p^2 \cos^2 \alpha + n^2 \sin^2 \alpha}} \\ &= \frac{(n^2 - p^2) \cos \alpha \sin \alpha d\alpha}{\sqrt{p^2 \cos^2 \alpha + n^2 \sin^2 \alpha}},\end{aligned}$$

and

$$\begin{aligned} dH &= \frac{[2p^2 \sin \alpha \cos \alpha - 2n^2 \cos \alpha \sin \alpha] d\alpha}{2\sqrt{p^2 \sin^2 \alpha + n^2 \cos^2 \alpha}} \\ &= \frac{(p^2 - n^2) \cos \alpha \sin \alpha d\alpha}{\sqrt{p^2 \sin^2 \alpha + n^2 \cos^2 \alpha}} \end{aligned}$$

Considering these differentials of forces between two confined molecules as thus required for moving one molecule from one preceding situation to a succeeding one, we obtain, by summation for the arc  $i i_1$ , the forces required for removing one molecule from  $i$  to  $i_1$ , and the total force of the pressure on the bearings during that movement. Calling the initial angle (namely, that included between the radius drawn to the point of first contact and the vertical)  $\alpha_1$  and the angle at the point  $i$  of release of contact 0, and summing between the limits of  $\alpha_1$  and 0, we have:

$$\begin{aligned} \int dC_s &= \int_{\alpha_1}^0 \frac{(n^2 - p^2) \cos \alpha \sin \alpha d\alpha}{\sqrt{p^2 \cos^2 \alpha - n^2 \sin^2 \alpha}} \\ \int dH &= \int_{\alpha_1}^0 \frac{(p^2 - n^2) \cos \alpha \sin \alpha d\alpha}{\sqrt{p^2 \sin^2 \alpha + n^2 \cos^2 \alpha}} \end{aligned}$$

The integrals of these two formulas will certainly be the equations 1 and 2, since these were the basis of differentiation. Introducing, therefore, the upper and lower limits, we shall obtain the total force consumed in rolling one molecule and the pressure caused by it. That is to say:

$$\begin{aligned} C_s &= \sqrt{p^2 \cos^2 0 + n^2 \sin^2 0} - \sqrt{p^2 \cos^2 \alpha_1 + n^2 \sin^2 \alpha_1} \\ &= p - \sqrt{p^2 \cos^2 \alpha_1 + n^2 (1 - \cos^2 \alpha_1)}. \quad \text{Now, put } p = en, \\ &= en - \sqrt{(e^2 - 1)n^2 \cos^2 \alpha_1 + n^2} \\ &= n [e - \sqrt{(e^2 - 1) \cos^2 \alpha_1 + 1}] \end{aligned}$$

and

$$\begin{aligned} H &= \sqrt{p^2 \sin^2 0 + n^2 \cos^2 0} - \sqrt{p^2 \sin^2 \alpha_1 + n^2 \cos^2 \alpha_1} \\ &= n - \sqrt{p^2 \sin^2 \alpha_1 + n^2 (1 - \sin^2 \alpha_1)}. \quad \text{Now, put } p = en, \\ &= n - \sqrt{e^2 n^2 \sin^2 \alpha_1 + n^2 (1 - \sin^2 \alpha_1)} \\ &= n - n \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1} \\ &= n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}]. \end{aligned}$$



These forces expending themselves on one molecule of iron of width  $dB$ , of thickness  $dh$  and length  $dL$  (Fig. 5), the mass ( $M$ ), which is found by multiplying its volume by the specific gravity ( $\gamma$ ) and dividing the result by the acceleration ( $g$ ), is:

$$M = \frac{dB \, dh \, dL \, \gamma}{g}.$$

The force required for, and the pressure on the bearings caused by, the rolling of one molecule will thus be:

$$C_{s_1} = n [e - \sqrt{(e^2 - 1) \cos^2 \alpha_1 + 1}] \frac{dB \, dh \, dL \, \gamma}{g}. \quad (3)$$

$$H_1 = n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] \frac{dB \, dh \, dL \, \gamma}{g}. \quad (4)$$

To determine the work of rolling, the force is to be multiplied by the distance through which it operates. This distance is the arc  $r\alpha$ ; but since  $\alpha$  remains constant for rolling through the same groove, the arc may be designated as a fixed fraction of the circumference, which we may call  $r \frac{\pi}{a}$ , and the work bringing one molecule into the desired situation accordingly will be as follows:

Rearranging the equation, and summing with reference to all the molecules of iron coming under work, we have:

$$H_m = n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] \frac{dB \, dh \, dL \, \gamma}{g} \frac{r \pi}{a}.$$

$$H_m = \frac{r \pi \gamma}{a g} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] \iiint dB \, dh \, dL.$$

The thrice-repeated summation of  $dB$ ,  $dh$  and  $dL$  represents that part of the ingot to be rolled which is marked by the decrease  $\frac{h}{2} + \frac{h}{2}$  in height (Fig. 5). Thus,

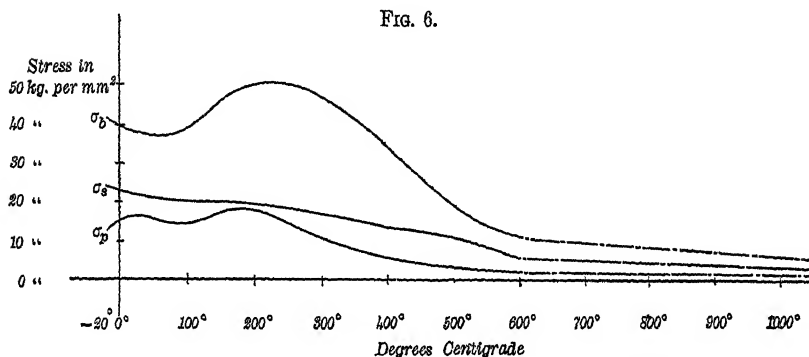
$$B \, h \, L = \iiint dB \, dh \, dL,$$

and the total energy expended in the work of rolling is therefore:

$$H_m = \frac{r \pi \gamma}{a g} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] B \, h \, L. \quad (5)$$

All the quantities on the right hand of this equation are known in any given case except  $e$  and  $n$ ; both of these are functions of the chemical composition and also the temperature of the material rolled. Moreover, the factor  $e$  (friction) depends upon the normal pressure. As to their exact value, experimental data are not now at our disposal. The intermolecular force of attraction is, however, approximately determinable by reason of the researches carried out by the technical *Versuchsanstalt* of Charlottenburg in the year 1890, published by M. Ledebur in the *Zeitschrift d. Vereins Deutscher Ingenieure* in 1896. But these researches do not furnish precise particulars,

FIG. 6.



$\sigma_p$ , Stress at the beginning of the proportional elongation.

$\sigma_s$ , Stress at the end of the proportional elongation, or the first stage of local elongation.

$\sigma_b$ , Stress at the moment of maximum loading.

DIAGRAM CONSTRUCTED FROM A TABLE IN THE *k. techn. Versuchsanstalt*  
OF CHARLOTTENBURG.

since the investigators examined the materials from a standpoint quite different from that of the present inquiry. They were trying to settle the conditions of safety of iron structures at high temperatures, and hence they did not follow their investigations above 600° C. Up to this limit they determined the stress per sq. mm. (line  $\sigma_p$ , Fig. 6); also the stress at the end of proportional or beginning of local elongation (line  $\sigma_s$ , Fig. 6); and also at the moment of rupture. Knowing, however, that steel is fluid at 1500° C., at which point the stress may be looked upon as practically zero, we can, by completing the diagram, Fig. 6, neglecting the unknown formulas of the curves and continuing them gradually as descending straight lines,

obtain an approximate value of the intermolecular force of attraction ( $n$ ) resisting tension, which will be a point on the line  $\sigma_s$ , Fig. 6, showing the stress of the molecules at the beginning of local elongation. This force varies from 5 to 3 kgs. per sq. mm. within the limits of the usual temperature of rolling, namely, 800 to 1100° C. Its exact value, however, must be determined for each kind of steel by itself; but the results once obtained will be good forever, and will thus always be at the disposal of the constructing engineers.

The analysis of the second factor, namely, the chemical composition, will also be constant when, in the course of time, the chemical character of any steel used for special purposes shall have been precisely established. The factor  $n$  is to be determined separately for the material of rolls, angles, structural and spring steel, etc., but, once determined, will remain constant and valid the world over. This determining and registering of values will enable us to solve problems occurring in ordinary practice referred to the standard factors to be used, and to calculate with confidence the difference of height between the two grooves for motors of given capacity, or, inversely, to construct suitable motors for any number of grooves and for given changes of dimensions.

In the case above discussed the diameter of the roll is assumed to be known. Sometimes, however, it may be necessary to determine the proper diameter of the roll. But, irrespective of special cases of regarding the influence of the factors  $\sin \alpha$  and  $a$ , it is not difficult to see that for effectiveness and economy of power the best results will follow the smallest practical value of  $rn \frac{[1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}]}{a}$ .

And for a given diameter the diminution of  $\alpha$  increases, and the constant increase of  $a$  reduces the value of the formula. If at liberty to increase the diameter, we shall, it is true, increase also the factor  $a$ , but in a smaller degree, and at the same time we shall decrease  $\sin \alpha$  and reduce the value of the fraction. All such changes will have their practical limits. For instance, too great an increase in the diameter would make the rolls too heavy; on the other hand, decreasing the angle ( $\alpha_1$ ), etc., would involve a much larger number of grooves and a waste of heat

by radiation during the longer period of rolling. We should probably adjust the diameter of the rolls, in proportion to the reduction of thickness effected in two subsequent grooves, with regard to the experience already obtained in rolling-mills. It may be mentioned here that the influence of the factors referred to above is hardly to be improved in mills of the present construction. The roll diameter varies slightly; and hence the angle  $\alpha_1$  increases and the factor  $a$  decreases most invariably in the blooming and roughing trains, with a consequent large waste of power developed by the engine. With the system proposed by me in the paper referred to, the ratio might be permanently settled between roll-diameter and the desired decrease of thickness of the slab, and this ratio might govern the construction. For the present we accept the formula proposed by the "Hütte-Manual of Technical Sciences," which, however, has been but seldom attained in rolling-mills thus far. According to it,

$$d - e = 0.01 R, \quad (6)$$

$d$  being the thickness of the ingot before rolling,  $e$  the thickness after rolling, and  $R$  the radius of the roll. The other factors in our formulas,  $B$ ,  $h$ ,  $L$ ,  $\gamma$ ,  $g$ , are very simple, and require no explanation.

To determine the factors  $p$  and  $n$  referred to above, we think the following the most suitable method:

Let the work done in rolling be expressed by

$$Pvt = H_m, \quad (7)$$

in which  $P$  = force,  $t$  = time and  $v$  = velocity. Equating this value of  $H_m$  with that obtained in formula (5), we have:

$$Pvt = \frac{r \pi \gamma}{a g} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] B h L.$$

Hence

$$Pv = \frac{r \pi \gamma}{a g t} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] B h L. \quad (8)$$

In experiments for such determination, a train should be chosen in which only one or not more than two pair of rolls are moving, and the simplest section, that of a rectangular par-

allelogram, should be rolled. Taking for experiments billets of identical dimensions and material (each rolled from the same ingot), keeping all conditions exactly similar, taking indicator diagrams on the engine when unloaded and when running at different velocities, and allowing also for the variation in the waste of power through friction in consequence of variable pressure, we shall be prepared to roll the experimental pieces. The billets should be passed through the grooves, changing the revolutions of the engine for each billet, simultaneously taking diagrams for the determination of the power consumed. The results of these diagrams, collated with equation 8, above, will give, for the values of the unknown factors, the formula,

$$n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}].$$

Since the amount of power consumed in each case exclusively in changing the shape of the mass would remain constant, and only the space of time occupied in the transformation would vary, it follows that the values of the factors in the expression of effect on the left side of the equation will be different in every case; but this variation being carefully observed during the experiment would be known, and thus there would be a series of data for determining the factor

$$n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}].$$

These data can be checked, corrected and averaged. Namely, altering the velocities in each case of the series of experiments, and taking simultaneous diagrams of the engine, the equation will put on the following form, in which the values of  $t_1$ ,  $P_1$  and  $v_1$  are known by observation :

$$P_1 v_1 = \frac{r \pi \gamma}{a g t_1} B h L n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}]$$

Taking the term  $n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}]$  in connection with the values given by the researches of the *Reichsanstalt* of Berlin, there will be no difficulty in determining the value of  $e$ . I must here remark that I think it necessary to make more precise experiments for the exact determination of the tensile strength of red-hot iron and steel. As already mentioned, the researches of the *Reichsanstalt* did not serve the same purpose which we want to reach. Moreover, speaking generally, we do not think it always correct to refer either the

maximum resistance or the stress of rupture to the original area of section of the specimen, being well aware of the changing of dimensions by elongation in full length, and by contraction in some part during the investigation. We have made some special experiments showing that the load carried per unit of area in respective moment of contraction, after the maximum resistance, decreases step by step.

We think it accordingly necessary to make further experiments, bestowing much care upon the parallel and simultaneous measuring of alterations of dimensions, and also upon the time required for such alterations, together with exact measurement of temperatures and stresses. Thus will be obtained the proper value of the factor  $n$  for certain material used generally and of known chemical composition. For the sake of complete certainty, the very same material prepared by rolling in the course of the determinations previously proposed should be used in these experiments.

Of course these investigations will have to be made separately for each iron and steel of different chemical composition. But the connection between the chemical composition and the factors above mentioned once having been determined, as far as a law exists, they will be practically constant, and we can have them registered in tables for the use of those who would make calculations of dimensions of engines to any train, or of grooves for rolls when engines of given effect are at hand.

For the determination of the value of factor  $n$  there may be found a second way which we can follow in the course of the proposed practical experiments simultaneously with the determination of the value of  $p$ . This factor  $p$ , in accordance with our hypothesis, is the friction on the unit of surface in contact. It follows, then, that  $p$  is equal to the product of the pressure ( $C_s$ ) on the bearings multiplied by the specific factor of friction  $f$  divided by the area of surface in contact.

$$p = \frac{C_s f}{a B}. \quad (9)$$

In this equation the factors  $C_s$  and  $f$  are as yet unknown quantities. The value of pressure on bearings one can weigh directly with balance or spring, etc., interpolated in proper manner into the construction of bearings, and for the determination of  $f$  we can proceed by reasoning as follows:

Given the pressure on bearings ( $C_s$ ) corresponding to the unit of surface of the red-hot slab in contact with the roll, and continuing the other marks given above, the formula expressing the force of friction is  $C_s f$ , and the projection of it on the direction of movement is  $= C_s f \cos \alpha_1$ .

The differential of this force being the angle  $\alpha$ , the independent variable factor is:  $d(C_s f \cos \alpha) = -C_s f \sin \alpha d\alpha$ ; that is, the force which works on that part of the surface defined by  $da dB$ . Hence  $-C_s f \sin \alpha d\alpha da dB$  represents the amount of force of friction, as related to the unit of surface. The work of this force will be obtained (given the peripheral velocity  $s$  of the roll) by multiplying the above formula of the force by the path  $s dt$ .

The differential of the work of friction will therefore be:

$$- da dB C_s f \sin \alpha d\alpha s dt.$$

And by summation we arrive at the total work of rolling:

$$a B C_s f s t \left( - \int_{\alpha_1}^0 \sin \alpha d\alpha \right), \text{ or} \\ - a B C_s f s t (1 - \cos \alpha_1).$$

Equating this formula with the value of work done by rolling and putting  $st = L$ , we have:

$$\begin{aligned} & - a B C_s f L (1 - \cos \alpha_1) = \\ & \frac{r \pi \gamma}{ag} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] B h L. \\ & - C_s f = \frac{r \pi \gamma}{a^2 g} h n \frac{[1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}]}{1 - \cos \alpha_1} \\ & C_s f = \frac{r \pi \gamma}{a^2 g} h n \frac{[1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}]}{1 - \cos \alpha_1} \quad (10) \end{aligned}$$

In this manner we obtain two equations (9) and (10), for the determination of factor  $n$ . For determining the factor of friction,  $f$ , the practical experiments of rolling will be executed in such a manner that, in rolling successive slabs of equal dimensions, a certain continuous increase of pressure would be applied in the same groove, up to the point at which the rolls do not draw in the red-hot iron, but only slide on it, so that the static friction changes to dynamic friction, thus impeding

the motion. In this case all the work of the driving-engine is consumed by the friction:

$$C_s f s t = P v t$$

and,

$$f = \frac{P v}{C_s s},$$

in which equation all the factors on the right side will be known. At the same time the work of rolling becomes zero:

$$\frac{r \pi \gamma}{a g} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}] B h L = 0.$$

But as far as each of the factors in the equation is a real quantity, the whole can become zero only on condition that—

$$\begin{aligned} 1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1} &= 0, \text{ or,} \\ e^2 - 1 &= 0 \\ e &= 1 \\ p &= n. \end{aligned}$$

That is, when the drawing force transmitted by friction becomes equal to the attractive force of molecules. In all other cases,  $p$  being greater than  $n$ , the work of rolling will be positive, and will depend for amount on the numerical value of the difference.

This condition—the revolution of the rolls performing no useful work—will set the extreme limits of values for angle  $\alpha$  and factor  $a$ .

The values of  $p$  and  $n$  having been determined by the equations 9 and 10, one can control the results obtained by the direct method of tensile-strength tests of specimens.

#### EXAMPLES.

I. We take a slab 180 mm. by 180 mm. in section and 900 mm. long. It is to pass the groove between rolls of 300 mm. diameter, with a decrease of thickness of 20 mm.—these conditions being usual in practice. According to Fig. 7, with the given dimensions the angle  $\alpha_1 = 27^\circ 42'$  and the factor  $a = 6.5$ .

$$P v t = \frac{r \pi}{a} \frac{B h L \gamma}{g} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 \alpha_1}].$$



Let the intermolecular force of attraction per sq. mm. be 6 kg. at the temperature  $1050^{\circ}\text{C}.$ ; from the line  $\sigma_b$  of the diagram, Fig. 6, and its supposed continuation we shall require a force of 12 kg. to overcome it, then :

$$P v t = \frac{150 \times 3.14}{6.5} \times 2.52 \times 6 [1-1.28], \text{ if } e = 2,$$

$$P v t = - 306.8;$$

and assuming the (not unusual) peripheral velocity of 1.5 m. per second,

$$t = \frac{0.9}{1.5} = 0.6,$$

and the effect of the motor will be :

$$P v = \frac{306.8}{0.6} = 511 \text{ kg. m.}$$

II. Let us follow the same ingot passing the last groove. The diameter of rolls will remain 300 mm.; the width of the piece, 200 mm., and the length, 9000 mm., will be taken into account, the decrease of height being 2 mm. According to these dimensions the angle  $\alpha_1 = 8^{\circ} 39'$  and the factor  $a = 20.7$ , as shown in Fig. 7. The piece is to be rolled from thickness 15 mm. to 13 mm.

$$P v t = \frac{2 \times 0.02 \times 90 \times 7.8}{9.8} \times$$

$$\frac{150 \times 3.14}{20.7} n [1 - \sqrt{1 + (e^2 - 1) \sin^2 8^{\circ} 39'}].$$

We assume that the red-hot piece of iron reaches the last groove with a temperature not less than  $800^{\circ}\text{C}.$ ; then the factor  $n$  found on the line  $\sigma_b$  of the diagram Fig. 6 is 9 kg. per sq. mm.; taking the velocity of the ingot as 2.5 m. per second, the applied force will move at this rate, and taking this ratio proportional to the square of the velocity,

$$e = 2.5^2 = 6.25, \text{ and}$$

$$P v t = 2.86 \times 22.75 \times 9 [1 - \sqrt{1 + 38 \times 0.1507^2}] = - 210.8.$$

Assuming the peripheral (usual in such circumstances) velocity

FIG. 7.

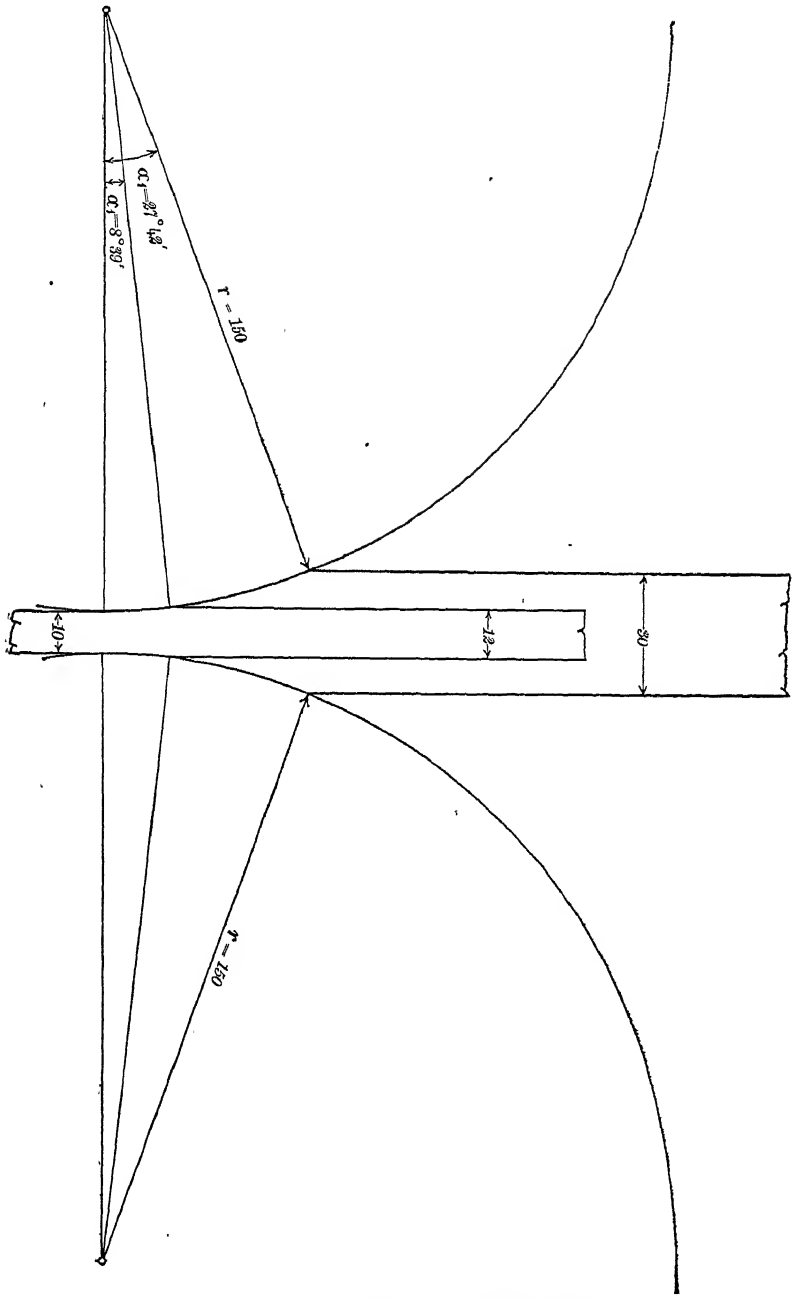


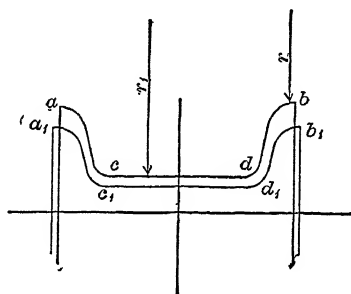
DIAGRAM ILLUSTRATING EXAMPLE II.

of 2.5 m. per second,  $t = \frac{9}{2.5} = 3.6$  seconds, and the effect of the motor is:

$$P v = \frac{210.8}{3.6} = 59 \text{ kg. m.}$$

I have thus far discussed only the section of a rectangular parallelogram. In rolling any other section, corresponding grooves having been turned in the surface of a cylinder, different points

FIG. 8.



of the grooves which do the rolling are at different distances from the axis of the roll; and in such cases the factor  $r$  in our formula will vary, even for the same groove. (See  $r$  and  $r_1$ , Fig. 8.) I think it very probable—I cannot speak with full certainty—that in such cases we shall make our calculations with some diameter of average value. Altering the radius of roll, the values of angle  $\alpha$  and of the factor  $n$  will also undergo trifling changes; but these changes may be eliminated sometimes by dividing properly the pressure (*i.e.*, decrease of thickness of ingots). There remains, however, the necessity of the determination of an average diameter. I take it here for granted that we shall find that dimension fixed by the distance of the center of gravity of the attacked sectional area from the middle line of the roll. I use the phrase “attacked sectional area” to mark the difference in area of sections deduced by comparison from the area of sections before and after passing the groove, and related accordingly to the upper and lower roll.

I must once more confess, in conclusion, that there are many experiments still to be made for these determinations, and they will very likely develop many new facts and views. The field is one in which expert investigators may both experience delight and win honor.

## The Condition and Action of Carbon in Iron and Steel.

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(New York Meeting, October, 1903.)

THE study of the condition and action of carbon in iron and steel is singularly complicated, because one has to consider, also, many contemporaneous reactions foreign to the one under investigation. Great difficulty is experienced in studying iron in the melted state, or during its various treatments in a solid condition, hence such an inquiry must be confined chiefly to a study of the original substance, of the conditions surrounding its treatment, and of the finished product. From a knowledge of these three things, we must form our own conclusions as to what takes place during the transition-period. Inasmuch as it is impossible to determine quantitatively the changes which take place in iron during the various processes of treatment and alteration, we are forced at once into the field of theory for an explanation of many of the phenomena for which we have to account.

A recapitulation of the recognized properties of iron and carbon, and of the conditions in which they exist in iron and steel, may be useful as an introduction to the discussion of the subject.

Chemically pure iron is soft, malleable and easily welded. It cannot be hardened or tempered, as these terms are understood in relation to steel. In contact with a magnet, it becomes magnetic, but loses this property when the magnet is removed. It is unfitted for castings on account of its lack of fluidity. Those properties of iron which render it valuable as a commercial product must, therefore, be primarily due to foreign substances contained in the iron; and the great variety of these properties is attained by changes in the amount and nature of these impurities, and by the physical treatment of the iron while it contains them.

The following properties of carbon make it a prominent factor

in iron and steel. It is a non-metal. It is very stable and has not been melted, even when subjected to the highest attainable temperature. It occurs in three different forms or modifications. It is believed to have a molecule containing a large number of atoms.

If we bring carbon in contact with molten iron, the iron may be said to act as a solvent and to dissolve the carbon.

Carbon may be present in solid iron and steel in any or all of four conditions, which have been called *hardening-carbon*, *carbide-carbon*, *temper-carbon* and *graphitic carbon*.

*Hardening-Carbon* is the form of carbon in hardened steel. It is also a prominent constituent in chilled iron. It cannot be separated from the iron in this form, and a direct study of its character is thus impossible.

The study of its effect on iron is most interesting. It adds hardness to iron in a striking manner, the degree of hardness being proportional to the amount of hardening-carbon present. When iron containing this form of carbon is treated with acids, the carbon unites with the hydrogen of the acid, forming volatile hydrocarbons. This action of acids upon hardening-carbon indicates either that there is a compound of iron and carbon which is broken up by the action of the acid, the carbon being left in the nascent condition to unite with the hydrogen of the acid, or that the carbon is in a state of sufficiently minute subdivision to be acted upon by the acid directly. This question will be further discussed under the head of "Theories."

*Carbide-Carbon* is carbon existing in a definite compound with iron, which can be separated and studied independently. This form of carbon predominates in unhardened steel, and occurs, to a greater or less degree, in all commercial forms of iron in which carbon plays an important part. If steel containing carbon is heated nearly to the melting-point and then allowed to cool slowly, the temperature decreases uniformly until it reaches about  $700^{\circ}$  C. Below this point the fall in temperature is retarded by an evolution of heat due to the formation of a carbide of iron ( $\text{Fe}_3\text{C}$ ). The more slowly the steel cools below  $700^{\circ}$  the greater is the amount of this carbide formed, and hence the greater the amount of heat evolved and the retardation in the rate of cooling. If the steel is again heated, the carbide disappears at a temperature slightly above that at which

it appeared in the process of cooling. This carbide is evenly distributed through the whole mass, and is of itself an intensely hard compound, and non-magnetic. If we dissolve iron containing it in cold, dilute hydrochloric or sulphuric acid, the carbide remains in the form of a grayish-black powder which, under the microscope, shows a silvery luster. It takes fire readily when heated; and I have frequently had a sufficient explosion, when determining it, to force the top from my combustion-crucible with a noise audible for some distance. The instability of the compound makes its accurate determination difficult. Ledebur, by treating steel with very dilute sulphuric acid, obtained a compound with a composition of 92.8 per cent iron and 7.2 per cent carbon. In a series of determinations performed on a sample of chilled iron, I obtained by treatment with dilute hydrochloric acid a residue which gave, upon analysis, results somewhat lower in carbon than those of Ledebur. These results were not very concordant, by reason of the fact that hydrochloric acid is apt to decompose the carbide partially, especially if heat is generated by the reaction. Results obtained by treating the same sample in a flask closed with a Bunsen valve, and using very dilute hydrochloric acid, were quite concordant, and very close to those of Ledebur.

*Temper-Carbon.*—This form of carbon existing in iron would be more familiar to the metallurgical world had it been named annealing-carbon, since it is obtained in the annealing of white iron. If a sample of iron, with all its carbon in the combined condition, were kept for some time at a temperature somewhat below its melting-point, there would be formed, in place of the white iron, a soft gray iron, easy to work, strong, and somewhat malleable. In fact, it would be the malleable iron of commerce. By this process the hardening-carbon would be changed to the so-called “temper” condition. If iron containing this form of carbon be dissolved in acids, the carbon is left behind in an amorphous condition.

*Graphitic Carbon.*—If the sample of iron used, as above described, in forming temper-carbon were heated for a much longer time, the carbon would continue to unite and form groups of carbon-atoms very much larger than the temper-carbon groups. Carbon thus grouped resembles in every way, and probably is, true graphite. The difference between the

temper-carbon and graphitic carbon is wholly due to a difference in the number of atoms in the molecule. Physically, the graphite has a crystalline appearance, but otherwise it generally resembles temper-carbon. There is no definite line separating these two forms of carbon.

*Carbon and Iron.*

When pure iron is heated to the highest temperature attainable it will dissolve, approximately, 6.67 per cent of carbon. If iron saturated with this proportion of carbon be cooled, the ability of the iron to hold carbon in solution decreases with the decrease of temperature, and the carbon begins to separate out as temper-carbon or graphite. The slower the cooling, the more complete is the separation of the carbon, and the more complete its development into graphite. As the graphite separates, the heat previously rendered latent in the solution of the carbon is given up. This liberated heat prolongs the cooling of the iron, and thus helps to produce more graphite. Not all the carbon will be found as graphite or temper-carbon, for in passing the degrees below  $700^{\circ}\text{C.}$ , some of it will unite with the iron as carbide-carbon, while some will remain as hardening-carbon. The proportion of each kind of carbon present will depend upon the rate of cooling. As has been remarked above, there is probably no definite line separating temper- from graphitic carbon. The same may be said of hardening- and temper-carbon. It is the custom to regard what will dissolve in boiling acids as hardening- and carbide-carbon, and what is left behind as temper- and graphitic carbon. But the most careful chemists have found that the amount of temper- and graphitic carbon thus determined varies with the strength of acid used. Temper-carbon and graphitic carbon are believed to differ only in the number of atoms to the molecule; and the same will be found to be true in regard to hardening-carbon and temper-carbon. The difference, then, between these forms of carbon lies wholly in the complexity of the carbon-grouping. The hardening-carbon has the simpler grouping, while that of the graphitic carbon is the more complex. The more complex the molecule, the stronger will be the required action of the acid which breaks it up, until a point is reached at which the carbon-grouping is unaffected by the action of the strongest acids.

The various processes through which iron is carried in the manufacture of steel and iron result in the formation of certain proportions of these different forms of carbon. The different properties of iron and steel depend largely upon the amount and the form of the carbon present in a given product. We are not satisfied to know that certain amounts of carbon present in iron produce different forms of iron or steel. We desire to know why this is so; and the nature of the problem shows at the outset that we must have recourse to theory for its solution.

### THEORIES.

The atomic theory regards matter as composed of infinitely small particles, called atoms, which are united to form molecules. The matter which our senses appreciate is produced by the aggregation of an innumerable number of these molecules. These atoms and molecules are in a perpetual state of motion, but always occupy a definite position in relation to one another.

The element iron, as well as the rest of the so-called true metals, is considered as having but one atom to the molecule. If we heat iron, the motion of its atoms becomes greater and greater and their attraction for each other less and less, until, finally, their coherence is so far overcome that the iron melts. In this state the individual particles are able to move freely about one another, but their coherence has not been completely destroyed. When iron is far below the melting-point, a definite amount of heat is necessary to produce a certain rise in temperature for each unit by weight of the iron. This amount is, approximately, the same for each degree of rise in temperature until the iron begins to soften. Above that point, the amount of heat necessary to raise the temperature of the iron one degree becomes greater and greater until the iron melts. This extra amount of heat produces no perceptible rise in temperature. It is absorbed by the iron and becomes, in fact, latent heat. The heat so absorbed serves to give a greater motion to the atoms, and thus being converted into motion is no longer perceptible as heat. When the iron solidifies, this motion is again transformed into heat, which prolongs the cooling of the iron. When sufficient heat has been applied to the iron to melt it, it becomes a liquid, and as such is subject to the laws



which govern liquids. In a liquid the particles are free to move around and about one another, but their coherence has not been completely overcome. A liquid is capable of dissolving many substances, and iron in a liquid state acts as a solvent, and as such follows the laws governing solutions. It is upon the substances thus dissolved by iron, and existing in it as impurities, that the commercial value of iron depends. There are a number of these dissolved substances, each playing its part in the formation of the varying conditions of commercial iron. The most important, in fact, the all-important, substance is carbon. When the solid carbon comes in contact with the melted iron, the transmitted heat from the iron gives to the carbon particles an increased motion, and the particles of iron gradually force their way among the particles of carbon, and the structure of the carbon is broken up. As the iron particles mix more and more with those of the carbon, the coherence of the carbon particles is finally overcome, and they move freely about with the iron particles. It must be remembered, however, that the attraction of these particles of carbon for one another still exists, although held in abeyance by the motion of the iron particles, and as soon as the heat is lessened, and the motion of the particles subsides, the particles of carbon will tend to re-unite with one another. Heat was rendered latent in overcoming their coherence, and heat will be given up when they re-unite.

*The Atomic Carbon Theory.*

The writer holds that this combined action of the heat and the molten iron upon the carbon decomposes or dissociates the carbon group into its atoms; that the carbon will remain in the dissociated condition only as long as the causes remain active which forced it into that condition; that it will return to its former state as soon as those causes are removed, unless prevented by some obstacle. The writer holds, however, that inasmuch as considerable time is necessary for the dissociation of the carbon structure by the combined influence of the heat and iron, if this heat could be immediately removed there would be no opportunity for the carbon particles to re-unite, but that they would be held by the iron in the form and position in which they were before the heat was removed, namely,

in the atomic condition. Let us imagine, for purposes of illustration, that the temperature of molten iron containing carbon could be instantly reduced, and the molten iron be changed to a solid so low in temperature that there could be no change in the condition of the carbon. In the molten condition the particles of iron and carbon were in very rapid motion, free to move in and about one another, but when the heat was instantly removed, this freedom of action ceased, and the particles of iron and of carbon, being so thoroughly intermingled, would prevent each other from re-uniting with their respective atoms, and the carbon would remain in the atomic condition. We have noted previously that if the carbon molecule was broken up heat would be rendered latent, and that when the atoms re-united this heat would be given up. If our supposition, that the carbon in our rapidly chilled iron was in the atomic state, is correct, then this carbon must still retain its latent heat. If it is proved, then, that the carbon in this hardened state does still retain latent heat, we shall have a strong proof that it is in the atomic state. That this is a fact, has already been proved. MM. Troost and Hautefeuille have shown that iron containing all its carbon in the hardening state gave a greater amount of heat, when treated in a calorimeter with mercuric chloride, than did pure iron under the same conditions. Osmond dissolved hardened and unhardened steel in double chloride of copper and ammonium, and found a greater evolution of heat with the former. These experiments show that the heat which was taken up in the dissociation of the carbon into its atoms was retained in part, at least, by the carbon in its divided condition.

While it is manifestly impossible in actual experience to reduce molten iron instantly to so low a temperature, nevertheless our imaginary experiment is sufficiently approximated by the well-known process of casting iron against chills. Here the heat is so rapidly removed that a considerable part of the carbon remains in the so-called hardening-state, which is none other than the atomic state of our imaginary experiment.

A second indication that carbon in the hardening condition is really in the atomic state is found in the action of acids on iron containing this form of carbon. The hydrogen of the acids unites with the carbon and forms volatile hydrocarbons.

Carbon in its normal condition is very inactive in presence of acids. This indicates that carbon in hardened steel must be in the atomic condition, in order to unite so rapidly with the hydrogen.

A third argument in favor of this atomic theory is found in the manner in which hardening-carbon unites with iron to form the carbide  $\text{Fe}_3\text{C}$ , and with atoms of its own kind to form temper-carbon.

Let us imagine, for illustration, two bars of unhardened steel containing, say, 1 per cent of carbon. One of these bars is heated to a high temperature and cooled suddenly. It becomes hardened steel. The other is heated to a high temperature and held there until it becomes thoroughly annealed, that is, until all the carbon is changed to annealing-carbon or graphite. We have now two pieces, one having its carbon in the form of hardening carbon, the other containing only annealing carbon and graphite. In both of these bars, whatever carbide existed in the first place has been broken up by the high temperature. Now, if the bar of hardened steel is gradually heated, a carbide is formed at a comparatively low temperature. On the other hand, if the annealed steel is gradually heated, no carbide will be formed. In other words, the carbon in the hardened steel is in a form which readily unites with the iron to form a carbide, while the carbon in the annealed bar is incapable of so uniting. The same results follow if both bars are heated to  $700^\circ$  and allowed to cool slowly. In the annealed bar no carbide is formed, while in the hardened piece the carbon readily unites with the iron to form the carbide. It will be readily seen, then, that the carbon must be in a peculiar condition when it is brought to the temperature at which the carbide is formed. Carbon and iron are not sufficiently opposite to have any great attraction for each other; and, if they do unite, they will most likely do so in the atomic condition. When we first heated our samples of steel, the carbon was changed to the atomic condition by the high heat and by the action of the iron particles. When the first bar was suddenly cooled, no opportunity was given to the carbon to resume the more complex form. The carbon is, therefore, held in the atomic condition in the hardened steel, and is, moreover, ready to prove its presence in the atomic condition, by uniting readily with the

iron to form a carbide when the bar is heated. If, instead of plunging the bar into cold water, we had plunged it into a metal bath heated to  $700^{\circ}$ , the atomic carbon would be caught as before, for by the sudden drop through the degrees of temperature at which annealing takes place, the atomic carbon is robbed of the opportunity to assume the more complex form. Hence, the carbon arrives at  $700^{\circ}$  still in its atomic state, and ready to form a carbide as the temperature falls through the degrees below 700. But once let the iron be kept at the temperature at which the atoms of carbon unite to form the molecules of temper-carbon, and the atomic state of the carbon is lost, together with its capacity to unite with iron; hence, the annealed steel was found quite incapable of producing the carbide when reheated. In it the carbon-atoms, combined into complex molecules, can be changed from their inability to form a carbide only by being heated to a temperature sufficiently high to break up the complex molecule and secure the atomic condition.

These three proofs confirm the theory that the carbon in melted iron and in the hardening condition in solid iron is in the atomic state. Upon this theory many of the problems which before seemed unsolvable are now readily explained. I attempted early to apply this theory to the solution of that much debated question, the reason for the hardness of hardened steel.

### *Theories of Hardening.*

There are at present two theories held by two distinct groups of scientists to account for the hardening of steel. These are the "carbon-theory" and the "allotropic theory."

Very briefly, the carbon-theory holds that the hardness of hardened steel is due to the fact that the carbon exists in a hard state at a high temperature, and that it is retained in this state by sudden cooling. This condition exists only at a high temperature, and becomes unstable during slow cooling; and at a lower temperature the carbon changes into the carbide found in unhardened steel. Sudden cooling prevents the formation of the carbide and, retains the carbon in the cooled steel in the same hardening state as that which existed at the high temperature.

The allotropic theory declares that the action is indirect, and that, above the critical temperature, the iron passes spontaneously into a special, strong, hard, brittle, allotropic state, which is unstable below that range, but that inasmuch as the passage from one state to another is relatively slow, the iron may be retained in the hard form by sudden cooling. The theory admits that the hardness is dependent on and proportional to the carbon present, and that no carbonless iron presents anything like the hardness of high-carbon steel.

So many eminent scientists are arrayed on each side of this question as to induce the belief that there must be truth in both views. The carbon-theory appeals to the mind because it gives a direct, tangible reason for the hardness. The chief argument against this theory lies in our inability to conceive of a substance so hard that 1 per cent of it, when mixed with 99 per cent of a substance as soft as iron, could produce the hardness of hardened steel.

*Field's Theory as to the Action of Carbon in Hardening Steel.*

An exhaustive study of the nature and action of carbon in steel, together with a careful consideration of the present debated theories, have led me to believe that the hardness of steel must be attributed to the co-operation of three distinct causes: first, the hardness of the carbon-atoms; second, the position of the carbon-atoms; and, third, the mutual attraction between the separated carbon-atoms.

1. *The Hardness of the Carbon-Atom.*—The objection to accepting this as the only cause lies, as has been said, not in its nature, but in its inadequacy. It is indeed difficult to imagine that the small amount of carbon, however hard it may be, can serve to harden the large amount of iron into steel. But it is evident that the presence of the hard carbon will materially aid in this process. The carbon that exists in steel is, without doubt, in a state of extreme hardness. The diamond is much harder than graphite; and it is probable that the diamond is composed of molecules which have fewer atoms to the molecule than have those of graphite. If we can assume that carbon in steel has an atomic grouping even less complex than the diamond, it is evident that the mere presence of such carbon in

the steel is a hardening factor of no small influence. And if, as I am thoroughly convinced, the carbon exists in steel in its atomic state, then the maximum of hardness may be reasonably assumed.

2. *The Position of the Carbon-Atom.*—Little need be said in explanation of this second reason. It assumes that the atoms of carbon are much smaller than those of iron, and that when the iron is in the molten state these atoms of carbon are thoroughly intermixed with the atoms of iron. If the iron were suddenly chilled, these atoms of carbon would remain between those of the iron and give a denser structure to the iron and a greater power to resist penetration or abrasion.

3. *The Attraction Between the Atoms.*—I have been led by a conviction of the insufficiency of accepted theories to go beyond the reasons advanced by advocates of these theories, and to search for a cause more potent than any as yet advanced. The allotropic theory, which seeks to explain the phenomenon by assuming the iron to be in an allotropic form, has two difficulties to overcome. The first is the fact that evidence indicates that iron has a monatomic molecule. While some allotropic forms are considered to be made up by a different arrangement of molecules, these molecules have in each case been composed of more than one atom. The second is, that if the allotropic form is due to a union in some unexplained way of carbon with iron, this union must exist at a temperature at which the carbon is in the free atomic state. These two considerations render the allotropic theory untenable, and led me for a long time to hold to the intense hardness of the carbon-atom as explaining the hardness of hardened steel. But I was finally convinced that the carbon-theory presented but a fraction of the truth. It states a cause, it is true; but there must be other and stronger causes, combining with the mere hardness of the carbon-atom. Hence the theory herein set forth, which differs from both of those described above.

The law governing solutions threw the first light upon the matter. The phenomenon of carbon passing from a high-carbon steel to a piece of low-carbon steel when the two pieces are placed side by side and heated to a high temperature, but not to the melting-point, led me to believe that iron at a high temperature follows the laws of liquids as to diffusion. Why,

then, should it not, at the same high temperatures, follow the same laws in other respects? The atomic theory says that a solid mass is composed of a large number of small particles which are in a constant state of motion, and that these motions are held in a fixed relation to one another. In liquids the particles are free to move in and about one another, but the attraction between them still exists. Let us apply this to carbon and iron in steel. The carbon particle in hardened steel we will suppose to be held in the atomic state. Let the steel be heated to a high temperature and these carbon-atoms will be in free motion, as is indicated by their passing from one piece of steel to another, and by their combining to form annealing-carbon and the carbide  $\text{Fe}_3\text{C}$ . Yet there still exists an attraction between these particles of carbon. Carbon as it exists under natural conditions has a very complex molecule, the attraction between the atoms of which is so great that heat alone, however high, has never succeeded in separating them. At a high temperature, in iron and steel, the heat so increases the motion of the carbon-atoms that the particles of iron move freely about with those of the carbon, but do not wholly destroy their attraction for one another. If, now, the heated steel, with the particles of carbon in the atomic condition, and with the attraction existing between these particles, be suddenly chilled, the carbon particles are held separated and in a fixed and unalterable position by the iron. But the attraction between the particles distributed broadcast through the iron acts as a network of forces, binding the iron particles together into a medium, dense and hard, which we know as hardened steel. This attraction acts only through short distances, as the nature of hardened steel shows; for such steel frequently has a high tensile strength, but never any great elongation. The attraction cannot be estimated, because it is impossible to tell how many atoms exist in a carbon molecule. This number, however, is known to be great; and it would not be difficult to imagine, from the complex nature of the molecule, that in 1 per cent of carbon there would be many more atoms than in 99 per cent of iron. These atoms, distributed throughout the iron, bind it together with a strong force of attraction, as hardened steel.

It is probable, moreover, that the atoms of iron, separated by

those of carbon, are also bound together by an attraction similar to that which binds the atoms of carbon. But this attraction, while it may be large in volume, is not nearly as strong as that between the carbon-atoms.

There is a difference in opinion as to whether chemically pure iron can show different allotropic modifications. Until that fact has been unquestionably proved, the writer prefers to consider that the increase in hardness and strength occasioned by heating and chilling low-carbon iron is due to the attraction of the carbon-particles and those of other impurities, combined with the increased attraction caused by the separation of the iron particles by these impurities, as above mentioned.

If we assume that absolutely pure iron can be hardened, there appears to be but one explanation of this phenomenon. When iron is in the unhardened state, the molecules, by a slow cooling, have arranged themselves in natural positions. The attraction existing between the particles is capable of extension under stress, as shown by the elasticity of iron under strain. This attraction, however, is limited, and, if it be exceeded, rupture takes place. When iron is subjected to heat, the motion of its particles becomes greater and the distance between the particles increases. Near the melting-point the attraction between the particles must be at its greatest tension. That is, the particles are in a condition in which but little more is necessary to break the bond which holds them together. They may be said at this temperature to be at their greatest wavelength. If they are chilled from this high temperature, and the heat is rapidly removed, they may be caught to a certain extent in the position of greatest tension. If we consider hardness as meaning resistance offered by a body to the separation of its particles, then the hardness would be increased by this process, providing it were possible to chill the chemically pure iron with sufficient rapidity to prevent the natural rearrangement of its molecules. This condition would correspond to the increased hardness which iron exhibits when stretched to its elastic limit. It may also coincide with the extra hardness caused by cold rolling. In short, if chemically pure iron is subject to hardening, it must be so, because of the greater resistance to separation which its particles offer as they are forced and held farther and farther apart, whether by heat or



by mechanical means. This conclusion is borne out by the fact that the increased strength caused by heating and chilling, stretching to elastic limit and cold rolling, is always accompanied by a lower elongation, thus showing that the attraction between the particles has been previously stretched by one of the causes mentioned.

### *Summary.*

The first two reasons for the hardening-action of carbon given above are not wholly new. The third is original with the writer. It avoids the objections advanced by the adherents of the allotropic theory, in that it does not rely upon the hardness of, say, 1 per cent of carbon to produce the result, but does rely upon the strong attraction existing between the atoms. On the other hand, this theory meets all the criticisms of the carbonists, in that it does not assume any mystic condition of the iron wholly unexplainable by facts or theories. It does assume a condition in accordance with, and substantiated by, facts and theories, as above stated.

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## **The Sperry Vanning-Buddle.**

BY EDWIN A. SPERRY, JOHNSTON STATION, GA.

(New York Meeting, October, 1903.)

IN 1892 I tested an ore, which was peculiarly difficult to treat, on several concentrators then in general use, including the Frue-vanner, the Luhrig table and the Cornish buddle.

Owing to the character of the ore it had to be crushed to a very fine state of division, and the results obtained were far from satisfactory; a large proportion of the values was lost in the slimes, which indicated the necessity of a very careful sizing before dressing. Even after reworking some of the slime-tailings, the results, in my opinion, were not as good as those which might be obtained on a table designed more especially for the handling of slime.

Although the subject of slime-concentration has occupied the attention of many engineers who have threshed the field quite thoroughly, the results obtained in my work led to a careful

investigation of the forms and principles of the various concentrators, with a view of devising a table which would be efficient in this class of work in general practice.

A comparison of the principles of concentration and the adoption of the best type in each case led to the conclusion that if a vanning motion of some kind could be given to a table of the buddle type, better results could be obtained. For obvious reasons the buddle form of concentrator is the best for slime work, and with this fact as a basis, the problem of motion and mechanical construction presented itself, a study of which resulted in the adoption of a revolving table with permanent points of feed and discharge.

All vanning motions are good, but some are better than others, an axiom which is especially true of circular motion compared with reciprocating motion, be it side- or end-motion; consequently the circular motion was selected as the proper one to use.

To be the most effective, the circular vanning motion should have an adjustment of the size of throw as well as of speed, as then the value of the rotating motion of the table would be greatly increased.

Cases exist in which an adjustment of the slope of the cone-shaped top is essential to good results, more especially if this adjustment can be made while the table was in motion.

A summary of the chief points to be incorporated in the design of a concentrator, in order to obtain the desired results, is as follows:

1. Buddle form of table.
2. A rotating table with fixed points of feed and delivery.
3. A circular vanning-motion.
4. An adjustment of the size and of the speed of the vanning motion.
5. An adjustment of the speed of the rotating motion.
6. An adjustment of the slope of the top which can be made while the table is in motion.

Taking up these points in order, the Buddle table, as stated above, was assumed as a basis.

The second point, in connection with third, demanded considerable study, as it seemed that both motions would be best applied at the center of the table and beneath the top; also,

one motion would probably amount to about 200 per minute, while the other would probably not exceed one rotation in the same time; a combination which allowed of the running of the vanning motion either in a forward or backward direction. The backward motion was found to be more effective, as it seemed to hold the metallics up against the action of the wash-water better. The term "rotation" is used in this paper with reference to the revolving of the table on its center, and "revolution" for the small circular vanning-motion.

The form finally adopted consisted of a hollow shaft at the center which could be actuated independently for the slower motion, and a solid shaft extended up through this hollow shaft to carry the rapid vanning-motion.

At the upper end of this solid shaft, a section 4.5 in. in length was made having an eccentricity of 0.25 in., and to this section was fitted an eccentric sleeve having the same degree of eccentricity, which by turning on the shaft allowed of an adjustment of total eccentricity of from 0 to 0.5 in. Furthermore, by placing the eccentricities against, or with, each other, any circle of motion from 0 to 1 in. was obtainable. This arrangement covered the fourth point, so far as the size of throw was concerned. The speed-adjustment was made by the use of expanding pulleys or similar devices. Both the size of the throw and the speed of the rotating motion were found to be quite constant in the treatment of any one ore, and having once been determined by test they could be adopted as permanent. The slight fluctuations liable to occur in general practice were regulated by another adjustment described later in this paper.

Of the several forms of mechanism considered for the purpose of adjusting the slope of the top, the one shown in Figs. 1 and 2 was finally adopted.

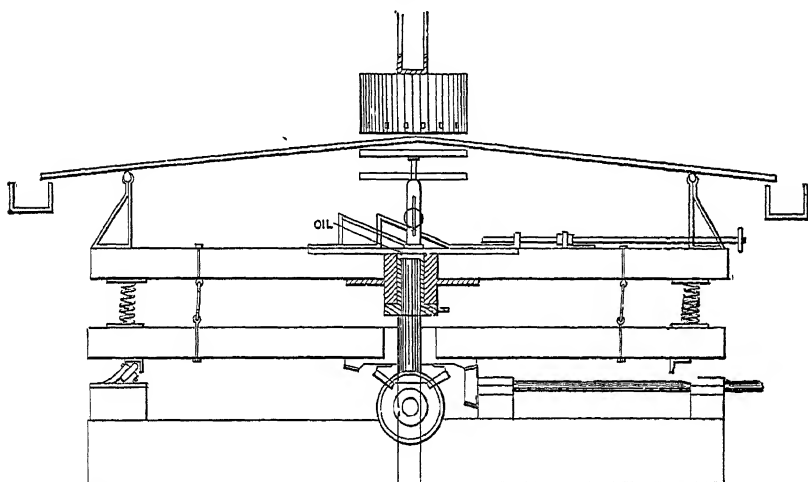
It was found by experiment that the surface best adapted to hold the slimes was smooth pure rubber, which gave the necessary elasticity for the slight increase of size when the top was raised to its extreme slope, and, if well stretched on the top while flat, the slight stretching on the increase of the slope made it very true.

In general construction, the first machine was built as shown in Fig. 1, but after running it was found that there was a ten-

dency toward an uneven motion on the lower wheel bearings, so the form shown in Fig. 2 was adopted. This latter form gave a steadier and smoother action, and is the one now used. In the following description, all references made relate to Fig. 2.

The heavy cast-iron base is designed to admit the hollow shaft, to which is attached a worm gear imparting the rotating motion to the table and giving the required reduction of speed. The upper end of the hollow shaft is in the form of a hub-flange, having attached thereto a six-armed spider-frame through which the rotating motion is transmitted. Within

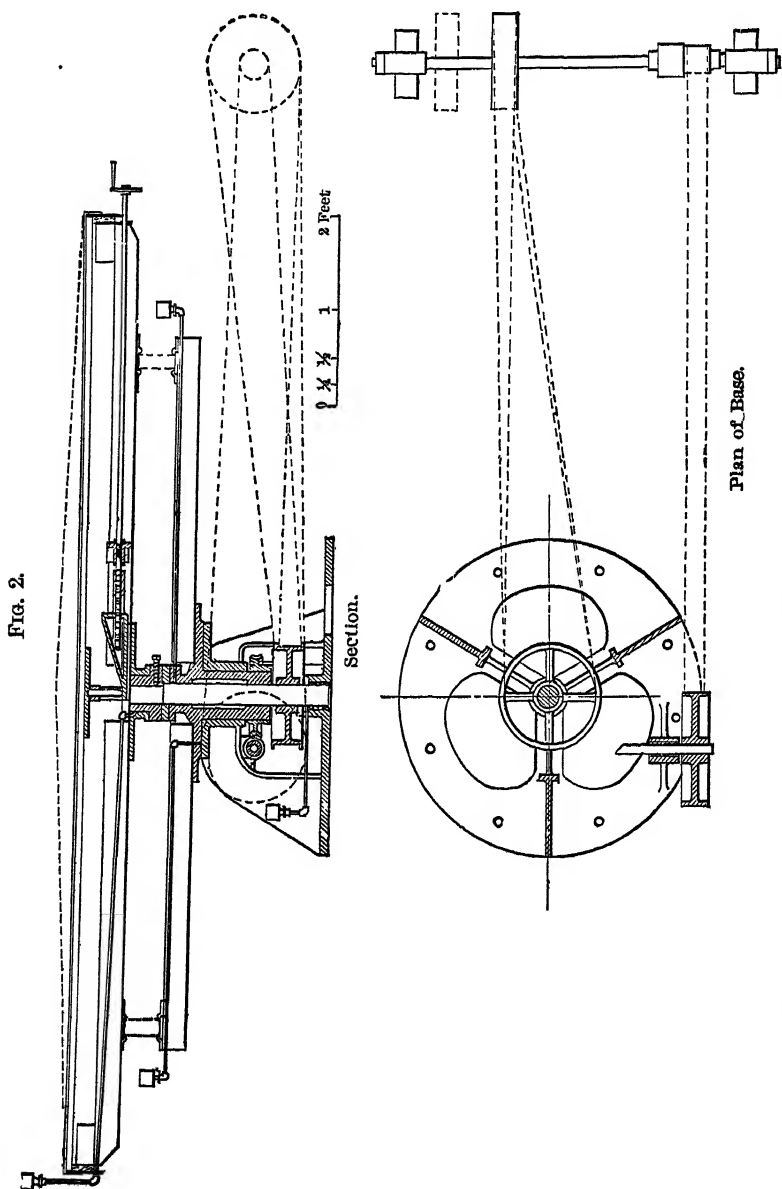
FIG. 1.



SKETCH OF THE FIRST SPERRY VANNING-BUDDLE, SHOWING THE METHOD OF ADJUSTING THE SLOPE OF THE TOP.

the hollow shaft is placed the solid shaft, the eccentric section of which extends above the hollow shaft. On this eccentric section is placed the eccentric sleeve, which is secured to the solid shaft by means of a heavy set-screw. Over this sleeve is placed a hub-flange, to which is attached another six-armed spider-frame, which in turn is connected to the lower spider-frame by means of spiral springs. When the hollow shaft is revolved, and with it the lower spider-frame, it transmits the motion through these springs to the upper frame. The solid shaft is free to revolve in everything except the eccentric sleeve. By revolving it imparts no motion to the lower frame,

but gives a small circular vanning-motion to the upper frame, the throw of which is dependent upon the position of the

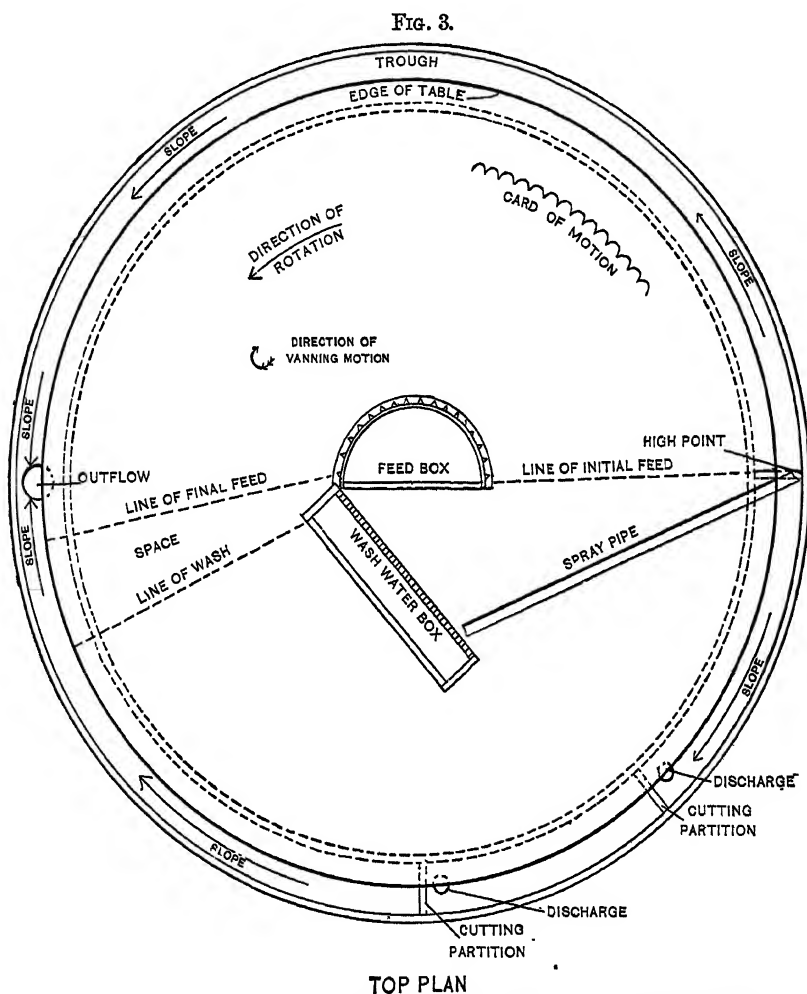


SECTION AND PLAN OF BASE OF THE LATER SPERRY VANNING-BUDDLE, SHOWING THE IMPROVED CONSTRUCTION.

sleeve. This motion is allowed by the elasticity of the connecting springs. In this manner the two motions are imparted

to the upper frame, and by this construction, it matters not in which direction the vanning motion is made to run.

On the upper frame is mounted a rim to support the top, which is composed of wedge-shaped boards placed radially,



PLAN OF SPERRY VANNING-BUDDLE, SHOWING DETAILS OF ARRANGEMENT.

the points resting on a plate supported by the device used for adjusting the pitch of the top. On this top is stretched the rubber cover which gives the required surface for working the slime.

As shown in Fig. 3, the feed-box is semicircular in shape

and is placed very nearly concentric with the table, the openings being on the circular side. The diameter of the feed-box is 18 in., which gives a peripheral length of feeding-space of 28 in.

Around the table is placed an annular trough which slopes to one point of outflow, the high point of the trough being located approximately in line with the back, and opposite the initial end of the feed-box. The outflow is located diametrically opposite the high point and opposite the final end of the feed-box. At the final end of the feed-box is placed a wash-water box which delivers water to the table in a gentle "washing" flow. This box is placed diagonally across the radial lines, the lower end reaching about half across the radius to a point almost directly behind the feed-box. From this point to the high point of the trough is placed a spray-pipe to deliver water under pressure to the surface, which it strikes with some force.

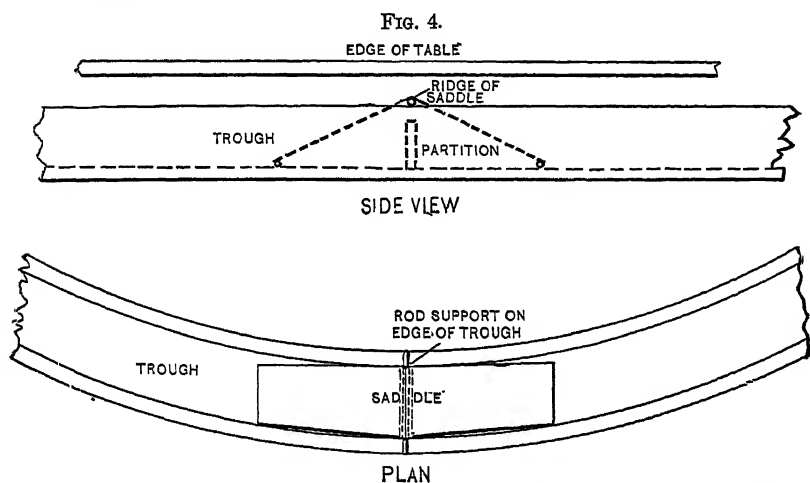
The general action of the table is practically divided into four stages: 1. The distribution of the pulp. 2. The stratification. 3. The washing of the "silica." 4. The washing of the metallics.

During the distribution of the pulp the flow increases from a width of 28 in. at the feed-box to a width of nearly one-half of the periphery of the table, or about 15 ft. on a 10-ft. table, which is the regular size in use at the present time.

There is a certain amount of stratification accomplished during the distribution. From the point where the table ceases to take the feed to the point where the wash-water first strikes the pulp is a small intervening space in which the stratification is finished and the pulp is prepared for the action of the wash-water, said action being carried on until the spray-pipe is reached where the washing is quite complete. The action of the spray-pipe is interesting, as it commences at once to wash the metallics of lighter specific gravity to the edge, in some cases making a very good quality of separated products. This feature has not been thoroughly investigated as yet, but from results that have been incidentally obtained there is quite sufficient ground for the belief that under certain conditions a practical separation can be made.

In general practice the pitch of the table varied from 0.75

in. to 1.25 in. per foot, depending on the character of the work. The speed of rotation was in most cases three-quarters of a revolution a minute, but in some special cases was as much as one rotation. The vanning motion was, of course, the most variable, being in different ores as low as 160 and as high as 210 per minute. The higher motions were used in cases where the specific gravities of the materials separated were widely different, and the lower, where they were more nearly the same. The size of the vanning motion varied from 0.25 in. to 0.75 in.



SKETCH OF THE SADDLE USED IN THE TROUGH TO SEPARATE THE CONSTRUCTION PRODUCTS.

The capacity was found to depend largely upon the condition of the pulp as it was fed to the table, being inversely proportional to its fluidity. In one case, 1,800 lb. per hour, of a peculiarly difficult ore, were successfully treated. Under ordinary circumstances, however, it is probably safe to calculate on 12 tons in 24 hours.

When the point of cutting between the headings and tailings has been determined, or if a middlings-product is desired, partitions are placed in the annular trough approximately at the points of cutting, and discharge pipes are fitted in the one or more compartments as the case may be. At these points, saddles are placed, as shown in Fig. 4, which by sliding backward or forward locate accurately the cutting point. This adjustment serves also to correct the slight fluctuations in the working of



the table mentioned above, which are common to all concentrating practice. The cutting space being from 4 to 10 ft. in length renders possible a very close separation.

A statement of some of the results obtained shows some very interesting facts.

Several tests were made with various tables on the zinc-sludge obtained from the Joplin district. Two of these tests are tabulated below, one made on a table of the Wilfley type and another on the Sperry vanning-buddle. The first results were not analyzed as closely as were the second, complete data being lacking; and on account of the limited cutting space there was no chance to make a series of numbers of concentrates.

TABLE I.—*Test of Joplin Zinc-Sludge Treated on a Table of the Wilfley Type. Nov. 13, 1901.*

Material.	Weight.	Zinc.	Saving.	Concentration.
	Pounds.	Per Cent.	Per Cent.	Ratio.
Zinc-sludge.....	164	6.6	.....	.....
Concentrates.....	18	34.62	57.67	9.1 into 1
Tailings.....	.....	3.10	.....	.....

TABLE II.—*Test of Joplin Zinc-Sludge Treated on the Sperry Vanning-Buddle. Oct. 26, 1901.*

Material.	A	B	C	D	E	F	G	H
	Weight Each.	Weight Combined.	Zinc.	Saving.	Zinc in All.	Zinc.	Concentration Actual.	Concentration Reduced to Terms of Table I.
	Pounds.	Pounds.	Per Cent.	Per Cent.	Per Cent.	Per Cent.		
Zn. sludge .....	163.25	.....	14.6	.....	.....	.....	.....	.....
No. 1 concentrates	7.9	7.9	66.5	22.0	.....	66.5	20.66	45.65
No. 2 concentrates	7.4	15.3	53.4	16.5	38.5	60.2	10.67	23.58
No. 3 concentrates	7.0	22.3	45.4	13.5	52.0	55.5	7.32	16.17
No. 4 concentrates	7.9	30.2	30.8	10.2	62.2	49.0	5.40	11.98
No. 5 concentrates	6.7	36.9	23.1	6.5	68.7	44.3	4.42	9.76
No. 6 concentrates	8.9	45.8	11.0	4.2	72.9	37.9	3.56	7.86

In Table II. column A shows the weight of the material on that line-column, B the combined weight of the concentrate on that line with those preceding, C the percentage of zinc in the material on that line, D the percentage of saving in the concentrate on that line, E the combined percentage of saving

of the concentrate on that line with all the concentrates preceding, F the percentage of zinc in that and in all preceding concentrates, G the proportion of concentration, and H the proportion of concentration reduced to the same basis of calculation as in the first test on the other type of machine. This last calculation of proportion was made on account of the wide difference in the values of the two samples, one containing 6.6 per cent and the other 14.6 per cent, or 2.21 times as much as the first.

A test was made to determine the fineness of the zinc and the following results were obtained:—

TABLE III.—*Condition of the Zinc-Sludge.*

Size of Material.	Weight.	Zinc.	Content.
	Per Cent.	Per Cent.	Per Cent.
Retained on 40-mesh sieve.....	21.0	3.58	6.3
Retained on 100-mesh sieve.....	25.5	2.75	6.2
Through 100-mesh sieve.....	53.5	16.8	87.5
Total.....	100.0	.....	100.0

The tests given in Tables I., II. and III. were made by William Russell, chief chemist of the Gold and Silver Extraction Co. of America, Ltd.

There are several very interesting deductions to be drawn from the two tests, one, taking the combined products more nearly equal to the value of zinc in the first test (in this case being the entire amount), a value of 37.9 per cent is obtained in the Sperry test as compared with 34.62 per cent in the Wilfley, with a concentrating proportion of 7.88 into 1 as compared with 9.1 into 1, and a saving of 72.9 per cent of the zinc as compared with 57.57 per cent.

Following out another line, taking the same percentage of saving, or 57.57 per cent, by interpolating between No. 3 and No. 4, which lies about halfway between the two, a value of 52 per cent is obtained as compared with 34 per cent, and a proportion of concentration of 10.84 into 1 as compared with 9.1 into 1. Other comparisons of equal interest may be made.

One trouble experienced was that as the material treated was not sized before dressing, some of the larger particles were caught by the water flow and dislodged, and in rolling along in

the current they cut a channel through the slimes which were already bedded and broke them up to some extent. As is clearly shown by the size tests, this difficulty could be easily avoided by sizing.

In April, 1901, a trial plant of two machines was installed at one of the mills at Aspen and operated part of two months. The results obtained demonstrated several interesting facts as shown in Table IV., on page 583.

The analyses given in Table IV. were made by the assayer of the mill at which the work was done.

The problem of the concentration of these tailings was confined almost entirely to the lead, the silver being so intimately associated with the gangue of dolomite that it was carried off with the tailings of the slime work.

The efficiency of the saving of slime values in this case especially was completely demonstrated in the following manner:— After several tons of concentrates had collected, hand-samples were taken from a number of places, which were dried and placed on a 200-mesh sieve. The entire quantity passed through the sieve with but two or three shakes. Another fact quite as pertinent was that on catching the overflow-water for a given length of time after it had passed through a series of five settling tanks, allowing it to become settled, decanting and drying, it was found, by comparing the resultant concentrates with the entire amount taken from the table in the same length of time, that about 10 per cent of the quantity of lead saved was floating off in suspension.

Other interesting features were brought out in this work, notably the very perfect stratification on the last quarter of the table, which showed a substratum of lead covered with a stratum of zinc, the latter from its character seeming to "shingle" the lead, so much so that it was rather hard to break it without using a pressure of spray considerably heavier than that usually used.

The table presented quite an artistic appearance, especially when the clear yellow of the zinc began to assume the blue tint of the lead, growing bluer until the lead gave its characteristic blue color, almost like a coat of blue paint. This circumstance indicated that the table might be of value in the separation of metallics, and a run on the middlings-product was made on May 15, but the product separated from the lead was not analyzed.

TABLE IV.—*Concentration Results Obtained at a Trial Plant at Aspen, Colo.*

Date.	Material.	Silver. Ounces.	Lead. Per Cent.	Zinc. Per Cent.	Saving of Lead. Per Cent.
April 22, 1901.	Feed, . . .	5.0	6.3	19.0	.....
	Tailings, . .	5.0	4.5	15.0	.....
	Concentrates, .	16.0	50.0	13.0	29.0
April 24th.	Feed, . . .	6.0	7.0	15.0	.....
	Tailings, . .	5.0	4.0	14.0	.....
	Concentrates, .	17.0	53.0	11.0	48.0
April 29th.	Feed, . . .	9.0	7.0	21.0	.....
	Tailings, . .	7.0	2.3	21.0	.....
	Concentrates, .	17.0	51.0	13.0	67.0
May 4th.	Feed, . . .	5.0	4.3	14.0	.....
	Tailings, . .	4.0	2.0	14.0	.....
	Concentrates, .	15.0	50.0	15.0	54.0
May 9th.	Feed, . . .	4.0	3.0	14.0	.....
	Tailings, . .	3.0	1.3	13.0	.....
	Concentrates, .	17.0	57.0	12.0	57.0
May 15th.	Feed, . . .	5.0	3.0	14.0	.....
	Tailings, . .	3.0	1.3	10.0	.....
	Concentrates, .	20.0	54.0	11.0	57.0
May 27th.	Feed, . . .	4.0	1.0	8.0	.....
	Tailings, . .	1.0	Trace	9.0	.....
	Concentrates, .	22.0	55.0	8.0	(?)
May 23d.	Feed, . . .	2.0	Trace	6.2	.....
	Tailings, . .	0.8	0.0	3.9	.....
	Concentrates, .	28.0	22.0	9.8	(?)

*Run of Middlings-Product.*

May 15th.	Feed, . . .	11.0	10.0	17.0	.....
	Concentrates, .	20.0	70.0	5.0	.....

*Runs of the Same Material on the Frue Vanner.*

May 3d.	Feed, . . .	6.0	5.0	16.0	.....
	Tailings, . .	5.0	3.3	16.0	30.0
June 10th.	Feed, . . .	2.2	3.2	14.0	.....
	Tailings, . .	2.0	2.0	14.0	38.0

NOTE.—The capacity of the Sperry Vanning-Buddle was from 10 to 12 tons per 24 hours, and that of the Frue Vanner was about 6 tons per 24 hours.

Another test of peculiar value and interest was made in May, 1901, on a telluride ore from Boulder county, Colo. Of the two statements of results given in Tables V. and VI., the first is of a run on the tailings obtained by concentrating the original ore on a table of the Wilfley type, with a view of maximum extraction by concentration, and following by treatment with the cyanide process. This original run showed an extraction of about 28 per cent, which left 0.78 oz. of gold in the tailings.

TABLE V.—*Results of a Test on Tailings from Table of the Wilfley Type.*

Date.	Material.	Weight. Pounds.	Gold. Ounces.	Value.	Saving. Per Cent.
May 7, 1901.	"Tailings,"	. 150	0.78	\$15.60	.....
	Concentrates,	. 5.5	19.04	380.80	89.50
	Tailings,	. . . . .	0.17	3.40	.....

TABLE VI.—*Results of a Test on Original Ore by the Sperry Table.*

Date.		Weight. Pounds.	Gold. Ounces.	Value.	Saving. Per Cent.
May 22, 1901.	Original ore,	. 150	1.22	\$24.40	.....
	No. 1 Concentrates,	10.	12.10	242.00	66.12
	No. 2 Concentrates,	12.5	2.36	47.40	14.81
	Total saving,	. . . . .	. . . . .	. . . . .	80.93
	Value of combined concentrates,	. . . . .	. . . . .	. . . . .	\$134.22

The length of the cutting-line renders it possible to make the concentrates of any desired value.

Many other results might be presented, but those given are sufficiently representative to convey a general idea of the work of the machine.

## The Metallurgy of the Homestake Ore.

BY C. W. MERRILL, B.S., MEMBER INST. MINING AND METALLURGY, ASSOC.  
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(New York Meeting, October, 1903.)

### I. THE PROPERTY.

The Homestake Mining Co. has acquired through consolidation the ground and equipment of the Father De Smet Consolidated Gold-Mining Co., the Deadwood-Terra Mining Co., the Caledonia Gold-Mining Co. and the Highland Mining Co., at and near Lead, Lawrence county, South Dakota, in the northern portion of the Black Hills. The company owns or controls 250 claims, comprising 2,616 acres and covering about 8,000 ft. along the strike of the lode.

At the surface there are several veins, of which three have united in depth, where the main vein ranges from 300 to 500 ft. in width. The 1,100-ft. level is the lowest at present. The rock of both walls is, so far as known at present, carboniferous slate, and the country is penetrated by a system of porphyry-dikes, and in some places capped with porphyry.

The output of the company up to January, 1903, has been, approximately, \$70,000,000.

### II. THE ORE.

The oxidized, open-cut ore is nearly all treated in the three mills on the northern part of the property, which are as follows: The Mineral Point (formerly the De Smet), of 100 stamps, the Monroe (formerly Caledonia), of 100 stamps, and the Pocahontas (formerly the Deadwood-Terra), of 160 stamps. A cyanide-plant, to treat the leachable portion of the tailings from these North End mills, has recently been installed and put into operation at Gayville, or Blacktail as it is now known.

The Homestake lower-level ore, which comprises the greater part of that being milled at Lead, may be described as a hornblende, garnetiferous schist or slate, which has been crushed and infiltrated with free silica and pyrites, the latter being

about 7 or 8 per cent of the ore, and comprising pyrite, pyrrhotite and traces only of chalcopyrite and arsenopyrite.\*

### III. MILLING.

The ore receives its first crushing in rotary breakers at the hoists, and this product varies in size from that of sea-sand up to rock having an extreme dimension of 4 in. From bins at the hoists at Lead, the broken ore is trammed to the three mills, *i.e.*, the Homestake and Golden Star, containing 200 stamps each, and the Amicus (formerly the Highland), in which there are 140 heads, making a total of 540.

From the mill-bins the ore passes to the mortar, which is of the now celebrated Homestake narrow pattern, where it is crushed between cast-iron shoes and dies, the weight of the stamp when equipped with new iron being 900 lb., the drop 10.5 in., falling 88 times per minute.

The screen is of the steel-needle slot-type No. 8, and the bottom of the screen-opening averages 10 in. above the top of the dies.

The long drop, high discharge and small area of screen openings produce an extremely fine pulp, about 80 per cent passing a 100-mesh screen, and it appears to me a most remarkable fact that under these conditions such a high stamp-duty is maintained, it being fully 4 tons per stamp per twenty-four hours.

This duty is possible only because: first, of the very favorable nature of the ore, the slate and pyrite crushing readily and the quartz being an excellent medium of attrition; secondly, of the large proportion of water used, being from eight to ten times the weight of ore crushed; and, thirdly, of the narrow mortar, which is only 12 in. wide at the lip.

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\* The standards used in this discussion are: the U. S. gold-dollar; the U. S. short ton of 2,000 lb. avoird., and the value of an ounce of fine gold, \$20.67. Percentages are given by weight, and not by volume. Sizings are classed as *coarse* (that portion of the sample which will remain on a 100-mesh screen; diameter of wire, 0.00433 in.; size of opening, 0.00575 sq. in.); *middles* (the material finer than the opening of the above 100-mesh screen, and coarser than the opening of the commercial 200-mesh screen as given below); and *finer* (the material which will pass such a 200-mesh screen; diameter of wire, 0.00216 in.; size of opening, 0.00312 sq. in.). A further subdivision of fines is also mentioned, which is based upon the granular, or angular, condition of one portion and the flocculent, or amorphous, condition of the remaining portion of these fines.

This very fine and thin pulp is in the most excellent condition for amalgamating, which process is conducted both inside the mortar and outside, on full-size plates in series (each 54 by 144 by  $\frac{1}{8}$  in.) to each mortar. The first of these is a copper-plate, and the other three are silver-plated copper, the weight of plating being 2 oz. per sq. ft. All the silver-plating is done at the works. The addition of the three silver-plates to each stamp-battery by Mr. Grier has proven one of the most valuable steps in the treatment of this ore, and has brought about an additional profit amounting to, approximately, \$250,000 during the year 1902, over and above what would have been realized from amalgamation had the outside plate-surface been only that of the one copper-plate—which, by the way, is considered ample in many of the large modern plants of the day.

In connection with amalgamation, the practice at the Homestake conforms, as far as conditions will permit, to the theory that the maximum results are obtained when the temperature of the water used in the batteries is low enough to exert the minimum influence on the minerals of the ore; and it is contended that the plate-yield proves the correctness of this theory.

It would be interesting to investigate the question of amalgamation and finer crushing in other gold-producing sections, particularly in South Africa, where the yield from this source is reported to be from 55 to 60 per cent, as compared with from 70 to 75 per cent at the Homestake. Perhaps finer crushing would not only greatly increase their amalgam-yield, but also reduce the values lost in their cyanide-residues. This seems the more likely for that country, because their slimes have been proved to have value sufficient for secondary treatment, whereas this has not yet been proved at the Homestake, where the advisability of sliming such a large proportion of the ore has been a debatable point, because the slimes here contain only from 85 to 110 cents in value per ton. But of this more will be said later.

The total cost of milling in the 200-stamp mills at Lead is, approximately, 40 cents per ton.

#### *Classification.*

We now have a pulp containing eight or ten parts of water to one of ore; and much of the latter is so infinitesimally fine



as to cause a visitor, who had watched an attempt to filter the slimes on a large scale, to say that, for an exemplification of the size of a molecule, he would advise the study of Homestake slimes.

The tailings as they leave the mill are sized, with the following result:

*Coarse* (remaining on 100-mesh), 22 per cent.

*Middles* (between 100- and 200-mesh), 18 per cent.

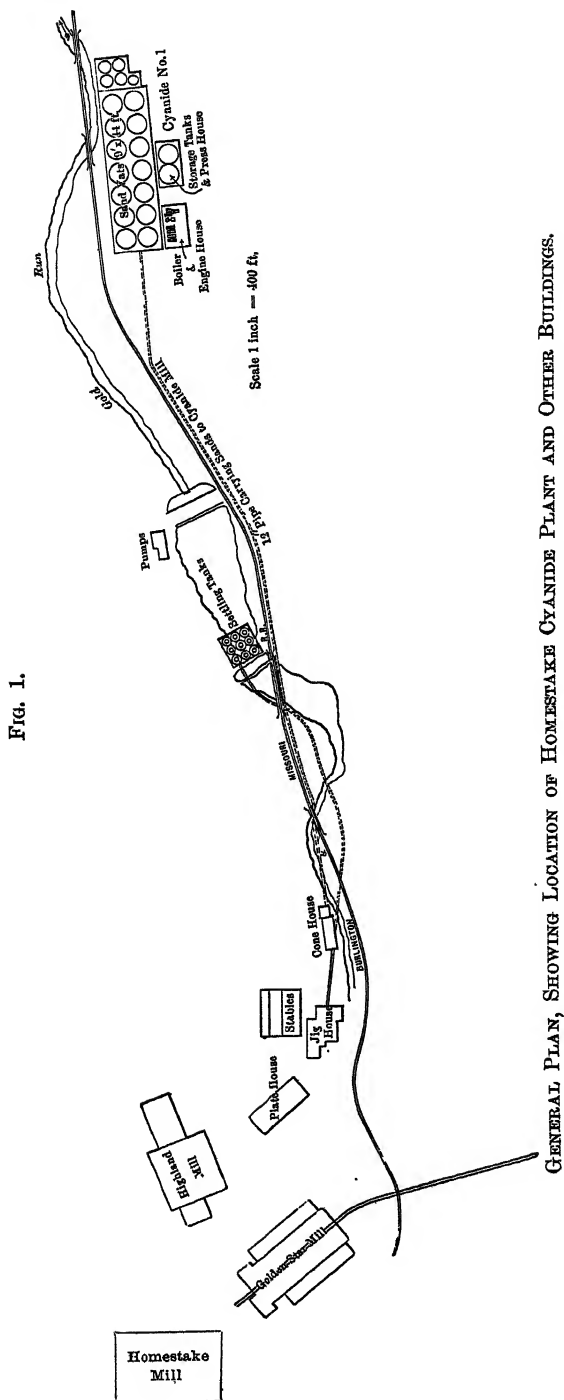
*Fines* (passing a 200-mesh screen), 60 per cent.

That is, 60 per cent of the particles issuing from the mortar have less than 0.00001 sq. in. of cross-section.

When the erection of the cyanide-plant had been determined upon, the question of a tailings-wheel to elevate the pulp and permit the location of the plant nearer the mills being under discussion, it was calculated that to elevate the tailings at a cost of about 2 cents per ton would cost the company, approximately, \$140,000, on the proportion of the material then blocked out in the mine which would be available for leaching. In other words, for every cent per ton which could be saved in the secondary treatment of the leachable material, the company would profit ultimately to the extent of at least \$70,000. Consequently, the plant was located, as shown in Fig. 1, about a quarter of a mile below the Lead mills; and the problems of transportation and of such classification as would permit the pumping-plant to return its former percentage of water to the mills, presented themselves. The latter has been met by the installation of the upper cone-house, where 12 gravity-settling cones, 7 ft. in diameter and with 50°-sides, throw off about half the water and, perhaps, one-fifth the solid matter, which latter is the very finest slime, of the following sizing, during 1902: *Coarse*, 0; *Middles*, 1.76; *Fines*, 98.24 per cent. The thickened slimes are, subsequently, settled out of this pulp, and a part of the water is returned to the mills.

From the bottom of the cones is drawn the thickened pulp, containing all of the leachable material and some of the slimes. This portion is transported by means of a 12-in. cast-iron flanged pipe on a minimum grade of 2.5 per cent, and with as few turns as possible, to the cyanide-plant.

The second step in the classification is carried out in the plant proper by means of 6 more gravity-settling cones, the



overflow from which, of a like composition to that of the first 12 cones, is conducted to a collecting-tank, whence it is drawn for the purpose of sluicing out the leachable material after its treatment has been completed. The average sizing of this second settling-cone overflow for 1902 was: *Coarse*, 0; *Middles*, 1.38; *Fines*, 98.62 per cent.

The under-flow from the second set of gravity-settling cones, which is now quite thick, passes to 24 sizing-, or hydraulic, classifying-cones, which carry a device for discharging the sand and introducing the water, patented by me. By its means the admission of water does not result in currents of varying velocity, which latter always interferes with uniform separation of slimes from granular material.

These sizing-cones complete the classification, which has been a difficult problem, first, because of the extreme fineness of the pulp, and, secondly, because I was determined to avoid double treatment, which entails a largely increased installation- and operating-cost, but which is necessary, unless a product be obtained practically free from slime.

The slime-overflow from hydraulic classifiers had the following sizing-average for 1902: *Coarse*, 0; *Middles*, 1.46; *Fines*, 98.54 per cent. As regards all slimes referred to, they will practically pass the 200-mesh screen, the middles being largely wood-pulp.

In fact, there is little doubt but that the importance of the most perfect classification possible will be recognized shortly as a vital consideration in the cyaniding of wet, crushed ore; and metallurgists will not follow the old German practice of *spitzkasten* and *spitzluten*, which are very imperfect machines as compared with a cone-classifier or sizer for separating granular from flocculent material. My judgment is that a scientific classification-system, by which all the granular or angular material may go to the leaching-vats, and all the amorphous portion to the slime-plant, will in the future be a feature in designing a plant on which the greatest care and experimentation will be put, and the highest grade of technical skill utilized.

#### *Cyanide-Treatment.*

By these three steps in the classification we have separated the pulp into non-leachable slimes, comprising about 40 per cent of the ore crushed, and practically all passing a 200-mesh

screen, and a direct-leachable product, amounting to, approximately, 60 per cent of the tailings, which, although very clean and free from mud, is still of a very fine texture,—as the following sizing-test, the average for the year 1902, will show :

*Coarse*, remaining on 100-mesh, 40.5 per cent. *Middles*, 100- to 200-mesh, 30.8 per cent. *Fines*, passing 200-mesh, 28.7 per cent.\*

While this fineness is notable, we find that, as the proportion of lower-level ore increases, we can treat an even finer product. A recent charge, containing as high as 40 per cent *fines*, maintained our normal leaching-rate of from 3 to 4 in. per hour throughout the treatment. This is undoubtedly due to the fact that the *fines* from the lower-level rock contain a greater proportion of angular or granular, and a smaller proportion of amorphous, hydrated or flocculent, material.

The leachable pulp, which contains from 10 to 12 per cent of pyrite, is now ready to go to the vats; and on the way lime is added in quantities varying from 3 to 5 lb. per ton. At first we tried adding this lime in the mills, as is done in Africa, but we found that the amalgamation was most seriously affected thereby; not only was the plate completely coated, weeks being required to get it back in proper shape, but the tailings-values were largely augmented. This result only emphasizes the fact that the process must fit the ore, and that attempts to make an ore fit a process are useless. This practice of adding lime to the battery is, according to my information, unanimously pronounced to work the best results in Africa, and to reduce the values in the slimes lost from amalgamation to half of what they are when no lime is used in the battery. In our case, however, we have demonstrated that the best results follow from crushing the lime wet into a running pulp which joins that from the sizing-cones, whereby there is less slacking and less loss of flocculent lime in the vat-overflow, *i.e.*, in the water which overflows the vat, the sand having settled out. Not only is it of distinct advantage to have the lime go into the tank in

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\* [A communication from Mr. Merrill, dated January 11, 1904, gives the information that an additional set of 16 gravity-cones has been installed, increasing the quantity leached by direct filling and percolation to approximately 70 per cent of the ore crushed in the mills. The average sizing of the material leached at the end of 1904 was :—*Coarse*, 34 per cent; *Middles*, 33 per cent, and *Fines*, 33 per cent. A larger proportion, and a finer average of the ore will be leached as soon as six additional 600-ton vats are installed.—R. W. R.]

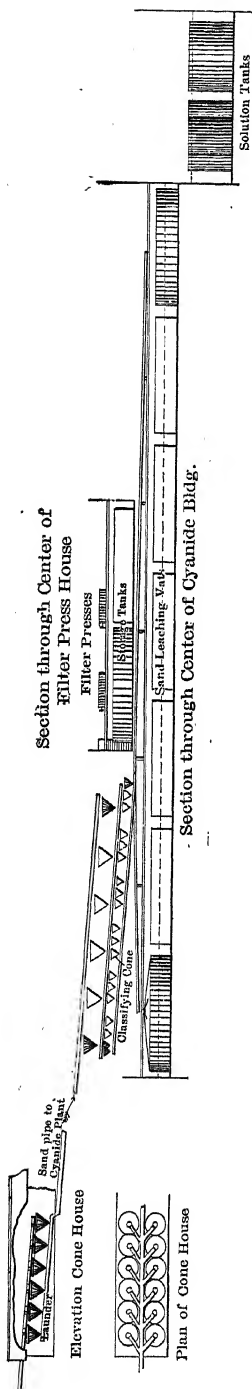
unslacked granules, but recent investigations are proving that the average size of these granules has an important bearing on the subsequent cyanide-decomposition and gold-extraction. This seems to be due to the fact that a low alkalinity, but one approximately constant throughout the leaching, is an important desideratum with the Homestake ore, on account of its considerable content of easily-decomposed sulphides. We are not, as yet, prepared to say what is the very best mesh-screen to use on our lime stamp-battery, but at present we are using a wire-screen, the opening of which is  $\frac{1}{8}\frac{1}{4}$  sq. in. In this connection it should be said that only the purest lime should be used, the magnesia in the ordinary domestic limestone being objectionable for several reasons.

The classified pulp and the lime having commingled, the mixture passes to the distributor, which is of the garden-sprinkler, or Butters and Mein type.

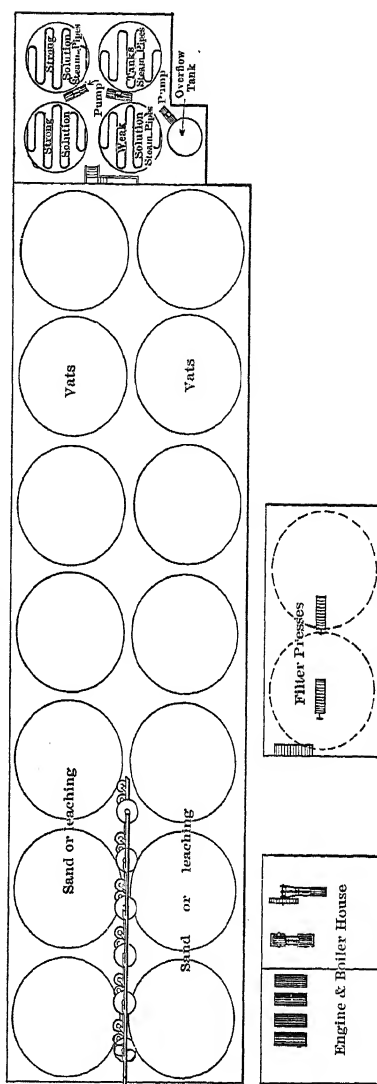
There are two distributors, one for each row of vats, hung from a carriage, which travels on a track, and the step of which rests on the top of the center-bottom discharge-gate of each vat, when the distributor is in operation. There are 14 vats, each 44 ft. in diameter, 9 ft. deep inside and holding 610 tons of sand. To fill one of these requires from eleven to eleven and one-half hours, which, with our equipment, permits of about five days' contact with solution, before it is necessary to recharge the vat. After filling, the drain-valve is opened, the top leveled, and the stronger of the two stock-solutions, of a strength of 0.14 of 1 per cent KCN, is run on. The contact with this solution, including frequent drainages for the purpose of drawing in air, is maintained for about three days. The air-contact is very important in Homestake ores, owing to the presence of pyrrhotite or iron subsulphide, which absorbs oxygen with great avidity, and which would greatly retard the dissolving action of the cyanide-solution were not large quantities of the essential oxygen introduced. The effluent solution during this period, having normally a strength of 0.10 of 1 per cent of cyanide, is run to the weak precipitation-tanks, of which there are two, each 26 ft. in diameter by 19 ft. deep, and holding 300 tons of solution.

After the three days' contact with strong solution, the weak solution, normally of a strength of 0.1 of 1 per cent KCN, is brought into the charge, and this contact is maintained for

FIG. 2.



Scale 1 inch = 74 feet  
Plan



# PLANS AND SECTIONS OF HOMESTAKE CYANIDE-WORKS, SHOWING STORAGE TANKS.

the remaining two days. The effluent solution from the charge during this period is run to the strong precipitation- or rather collecting-tanks, which are of the same size and number as the weak precipitation-tanks.

Fig. 2 shows the interior arrangement of the works.

After contact with the weak solution has been completed, wash-water is brought into the charge, and the washing continued until the effluent solution is down to 0.03 or 0.02 of 1 per cent in KCN and from 5 to 7 cents per ton in value.

The charge is now ready for sluicing, which operation is accomplished by two men, with 3-in. hose, in about four hours, using the slime-water from the overflow of the second settling-cones. The 4 side-gates and 1 center-gate afford ample facilities for the discharging. The last inch or so of the sand is sluiced with clear water under 75 lb. pressure through 1.5-in. hose; and the 8-oz. duck filter, under which is another of cocoa-matting, is washed clean. The vat is then filled with water, and is ready for the next charging.

*Precipitation.*—As stated above, the effluent solution resulting from the leaching with strong solution is run to the weak precipitation-tanks, and has a value of, approximately, \$2 per ton and a strength of 0.10 per cent KCN. When one of these weak precipitation-tanks is full, the stream is turned to the other, and the former is then ready for precipitation. It contains 300 tons of solution, which is brought into agitation by means of compressed air, and about 60 lb. of zinc-powder, in the form of an emulsion, is sprayed in during the agitation. The pump, which is of the compound, duplex, outside-packed, plunger-type, is then started, and the mixture pumped through two large filter-presses 36 in. sq., of the flush-plate and distance-frame pattern, containing 24 frames, each 4 in. in depth.

While the gold, silver and excess of zinc remain in the frame and on the cloth, the barren solution passes through the cloth and on to the weak solution storage-tank below (of the same size as the sand-vats), whence it passes again to the sand as weak solution. Its value has been reduced by this operation from \$2 to 5 or 10 cents per ton, being a precipitation of from 95 to 97.5 per cent. The efficiency of this method lies largely in the fact that the cloths of the presses are coated with about  $\frac{1}{8}$  in. of powdered zinc and precipitate, so that every particle of solution, having to pass through the cloths, gets a molecular contact

with the fine zinc, which is true of no other precipitation-process. The presses are run without opening for a month, at the end of which the press-gauges indicate about 10 lb. pressure, notwithstanding the fact that they then contain about a ton of precipitate worth, say, \$50,000; they are then cleaned up by two men in about six hours, including the putting together with new cloths. Figures covering the labor of cleaning up \$50,000 from zinc boxes and from electrolytic precipitation would form an interesting comparison.

We will now return to the effluent solution, resulting from the contact of the tailings with weak solution during the latter part of the leaching. This is run to the strong-solution collecting-vats. When these are filled, they are strengthened to 0.14 per cent KCN and pumped directly, without precipitation, to the strong-solution storage-tank, of the same capacity as the weak storage, whence it goes on to the early treatment of the charge, as before mentioned. Its value is from 30 to 50 cents per ton. It will thus be seen that the strong solution of one day becomes the weak solution of the next day, and that the values are all accumulated in the weak precipitation-tanks. The strong solution thus has an approximately constant value, that is to say, only one-half of the total effluent solution is precipitated, the other half being of a constant low value.

*Refining Precipitates.*—As the refining of cyanide precipitates is of some importance, owing to the well-recognized losses taking place in the ordinary methods, which are from 2 to 6 per cent, a description of the process we use at the Homestake, in which the loss is less than 0.1 per cent, may be of interest.

The precipitate, after removal from the presses, is treated first with dilute hydrochloric acid in a lead-lined mixing-tank, equipped with a mechanical agitator, a hood, and a powerful exhaust-fan. After agitation and settling, the supernatant liquid is forced through a filter-press by air-pressure. Sulphuric acid is then added, agitation begun, and the mixture heated. It is then settled, and the supernatant solution put through the press, as in the case of the hydrochloric acid. Wash-water is then added to the mixing-tank and the whole mixture put into the press, where it is further washed. The aggregate value of the acid-liquors and wash-water flowing from the press is less than \$20, from \$50,000 worth of precipitate. A portion of this value is recovered from a large settling-



tank, into which the effluent solutions flow, and the remainder constitutes the only loss we have been able to find in this process of refining.

The resultant, acid-treated precipitate is then removed to a large steam-dryer, where a part of the moisture is expelled, but never all, and the precipitate mixed with litharge, borax, silica and powdered-coke. When thoroughly mixed, it is sprinkled with a solution of lead-acetate and the whole mass briquetted under a pressure of from 4,000 to 6,000 lb. per sq. in. The zinc having been removed, and the briquettes having been dried, a borax-slag develops upon the outer surface upon being charged to the cupel, and they fuse quietly, quickly and at a low heat, without dust or volatilization-losses. The lead absorbs the values, sinking to the bottom, and the slag is tapped off. All the slag having been removed, the lead is cupelled off as litharge, and the resultant metal, from 975 to 985 fine, is ready to run into bars. The cupel-slag and the cupel-bottom are then put through the blast-furnace, the lead-content of the slag reduces to lead, which absorbs the values, and is drawn from the lead-well in the usual manner. This lead is returned to the cupel at the next clean-up, the litharge from the cupellation goes to the next precipitate, and the blast-furnace slag is worth less than \$5 per ton.\*

The total cost of this refining amounts to less than  $\frac{1}{4}$  of 1 per cent; so that the Homestake Company realizes \$20.52 per oz. for its cyanide-gold, less the usual U. S. Assay-Office charges on doré bullion, and the expressage to New York. These charges amount to between 10 and 11 cents; and the net realization per oz. of fine gold precipitated is thus \$20.42 in New York exchange. A parting-plant is now contemplated, which will make a further saving in this connection and enable the company to turn out fine gold and fine silver.

#### TONNAGE, PERCENTAGE AND COSTS.

*Tonnage.*—The maximum monthly tonnage of this plant—which is ascertained by placing cubic-foot boxes in many parts of various vats, determining the dry-weights per cubic foot of sand and averaging a large number of such determinations,—was attained in October, 1902, when 40,236 tons, or 1,298 tons

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\* I have applied for patents covering this process, which was first carried out experimentally during the latter part of the year 1900.

per day, were treated.\* This gives to the Homestake Company the largest sand-treatment cyanide-plant in the world; the next largest being, to the best of my knowledge, that of Simmer & Jack in South Africa.

*Percentage.*—As a comparison of the various assay-determinations and valuations with the bullion produced is always of interest, the following figures for the last half of the year 1902 are given :

*Extraction.*—The extraction, as shown by the difference between charge and residue-assay multiplied by the tonnage, was \$292,579.

*Precipitation.*—The precipitation, as shown by the difference between assays of unprecipitated and precipitated solutions multiplied by the solution-tonnage, was \$301,233.

*Gold in Precipitates.*—The amount of gold in precipitates, that is, the assay-value of the precipitate sampled upon removal from the presses, was \$302,895; the gold-value of bullion shipped, \$307,635, and the silver-value, \$2,874.

The average percentage recovered in bullion by the treatment for these six months is 74.7 per cent.

This is not as high a percentage of bullion as should be recovered from a porous or oxidized ore, or one in which the values are along cleavage-planes; but, in view of the facts that such a high percentage is recovered by amalgamation, that the values are very finely disseminated in the Homestake ore, and that the tailings are very low-grade, we feel, and all our tests so far have verified our conclusions, that it is the economic percentage, yielding the maximum net profit.

Many tests and experimental runs of the plant, looking toward a greater net yield, have been made, covering longer treatment, stronger and weaker solutions, extra oxidation with sodium and barium dioxide, and other similar reagents, varying alkalinities and alkaline reagents, etc.† The question of separate treatment of concentrates and coarse sands has also been investigated, all with negative results. The conclusion I have reached in regard to this point is that, even if a higher net yield could be realized by separate treatment, which is contrary to the results of all our tests, a much greater proportion of the

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\* Since this paper was written, another step in the classifying of the pulp has been added, with the result that this plant is now treating approximately 1,450 tons per 24 hours.

*fines* (passing 200-mesh screen) would have to be thrown off and wasted, entailing a serious net loss.

*Costs.*—As to operating-costs at the Lead cyanide-plant, the following are the averages per ton for the year 1902, during which the average value of the material treated was \$1.65 per ton :

Classification—Labor and Supplies,	. . . . .	\$0.017
Treatment :		
Cyanide,	. . . . .	\$0.152
Labor,	. . . . .	0.030
Lime,	. . . . .	0.022
Supplies,	. . . . .	0.005
		<u>\$0.209</u>
		0.209
Precipitation—Labor and Supplies,	. . . . .	0.026
Power—Labor and Supplies,	. . . . .	0.051
Water,	. . . . .	0.026
Assaying—Labor and Supplies,	. . . . .	0.013
Refining—Labor and Supplies,	. . . . .	0.006
Miscellaneous,	. . . . .	0.005
		<u>0.005</u>
Total,	. . . . .	\$0.353

As compared with the above, the lowest costs I have seen authoritatively stated for other plants are as follows :

City and Suburban, South Africa,	. . . . .	\$0.55
Geldenhuis Estate, South Africa,	. . . . .	0.605
Geldenhuis Deep, South Africa,	. . . . .	0.62
Robinson, South Africa,	. . . . .	0.62
Worcester, South Africa,	. . . . .	0.72

The African costs refer, of course, to operations before the late war between England and the South African Republic; but they are the only figures available to me, and I do not think they have been reduced materially since.

As regards the Homestake slimes, which are not at present being treated, their assay-value ranges from \$0.80 to \$1.10 per ton, which is very much lower than that of any slimes now being cyanided elsewhere, and does not offer much inducement to undertake their hydrometallurgical treatment as a whole. It is my judgment, however, that a material profit, greater than that attainable by any method of treating the whole bulk of slimes, may be recovered by concentrating these slimes and cyaniding the concentrates so obtained; and experiments on a working scale will shortly be undertaken along these lines.

## The Cyanide-Plant and Practice at the Ymir Mine, West Kootenay, British Columbia.

BY EDWIN C. HOLDEN, NEW YORK CITY.

(New York Meeting, October, 1903.)

IN making the original estimates for a report recommending the cyanidation of Ymir stamp-mill tailings, the writer was unable to find in any of the standard works on cyanide-practice certain data applicable to the local conditions; nor has he as yet seen quite the same practice recorded in the rapidly growing literature of cyanidation. To present these figures, together with a few novel features of local practice, the present paper is offered.

The ore treated is obtained from a fissure-vein of steep dip, occurring in a slate country. The ore-body is lenticular in plan, swelling from 4 ft. in width at one end to a maximum of 42 ft., and tapering irregularly to less than 18 in., where pay-values cease. In the wider portions of the fissure the ore is mixed with more or less horse-matter from the walls. The clean ore is white, massive quartz, with from 8 to 12 per cent of sulphurets; the latter being pyrite, sphalerite and galena, the predominance being in the order given. The precious metal tenor is variable, the gold more closely following the pyrite, the silver the galena. The present mill-supply is practically all from below the oxidized zone.

The unavoidable admixture of horse-matter and stope-filling with the ore as mined, affects more or less the character of the mill-feed, which sometimes contains from 10 to 25 per cent of slate. During the last three months of 1902, the mill-feed carried 0.3977 oz. gold and 1.903 oz. silver per ton of 2,000 lb., 2.65 per cent lead and 2.92 per cent zinc. The iron, unfortunately, was not determined.

The Ymir mill and mine have been fully described elsewhere;\* suffice it to say of the milling that the crushed ore,

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\* *Journal of the Canadian Mining Institute*, 1900. "The Ymir Mine and its Mill-Practice," by S. S. Fowler.

which will pass a 2-in. grizzly, is fed into sixteen 5-stamp batteries. The mortars are narrow; the stamps average 850 lb. and drop 6.5 in. from 98 to 100 times per minute. The height of issue is from 3.5 to 5 in., and the screens (diagonal slot) are No. 9 or 11, depending upon the height of issue. The crushing-capacity is 2.5 to 3 tons per stamp per 24 hours. There are no inside plates, and but one 56- by 144-in. apron-plate to each mortar. The saving on the plates was 61.9 per cent of the gold and 9.4 per cent of the silver.

The plate-tails from each ten stamps were, without classification, put over three Frue vanners. A slightly variable concentrate was made, averaging 20.6 per cent lead; 1.17 oz. gold and 12.5 oz. silver per ton; and, approximately, zinc, 12.9; iron, 23; and insoluble, 6 per cent. There was an extra smelting-charge on zinc in excess of 8 per cent; and it was a delicate matter to determine how high the grade of the vanner-tails could be raised with profit, so as to throw over the zinc and reduce the concentrate-tonnage. The gold-saving in the concentrates was 16, the silver 35.4, and the lead 42 per cent. The total saving in the stamp-mill was thus 77.9 per cent of the gold and 44.8 of the silver. These totals were higher before the tailings-plant was installed, when both battery- and vanner-work were necessarily closer.

The vanner-tailings during the quarter ending January, 1903 (and all the figures here given, unless otherwise stated, are for that period), assayed 0.0882 oz. gold and 1.051 oz. silver per ton; lead, 1.3; and zinc, 2.2 per cent. Of this material, 65 per cent would pass a 100-mesh screen.

The vanner-tailings being of comparatively low grade, I felt that I was practically confined to the adoption of the system of direct filling and percolation; and, as slimes-treatment was not at the same time to be provided for, the object was to treat as large a proportion of the slimes as possible with the sands; or, in other words, to make charges having the lowest practicable rate of percolation. This minimum rate is usually stated as 2 in. per hour.

It is also a current conception that a much larger proportion of slimes can be handled in an intermediately-settled charge than in a directly-filled one. To determine whether this possible difference in tonnage was important enough to justify the increased cost of installation and operation of a plant with

settling-vats was an important problem, which our experimental plant decided.

After the usual laboratory-tests, the experimental plant, consisting of two 35-ton leachers, 5 ft. deep, with solution-tanks, zinc-boxes, etc., was erected. One leacher was charged direct from a Butters distributor, which was fed by the bottom-discharge of a pointed box carrying the vanner-tailings from 20 stamps. The box got rid of excess-water and a little of the finest slimes. The second leacher was charged from a two-compartment box in which the vanner-tailings were settled, the sands being dropped from the compartments alternately as filled, and shoveled into the leacher as in ordinary intermediate filling-plants. We were thus enabled on a commercial scale to compare results of the two systems of filling with the same character of feed. Owing to structural difficulties in the experimental plant, which had to be squeezed into an unoccupied corner of the stamp-mill, there was not head-room enough to allow of feeding the settlers from the pointed box nor could we give the settling-boxes area enough to settle charges carrying the same proportion of fines as the direct-filled ones. A comparison of results from the treatment of 1,525 tons, in charges averaging 35 tons, is here given, screen-tests being 100-mesh:

Charged.	Fines. Per Cent.	Gold Recovery. Per Cent.	Silver Recovery. Per Cent.
Direct, . . .	49.0	74.1	48.0
Intermediate, . .	36.5	76.2	50.2

The average minimum rate of percolation in the direct-filled charge was 1.9 in. per hour. In the intermediately-filled charges the rate was quite variable, but the average minimum was over 3.25 in.

It is to be regretted that, for the reasons above given, it was not possible to make charges having a closer screen analysis. It seems, however, reasonable to conclude that, although the fines in a charge always yield a higher percentage of their values than the sands, if the proportion of fines in the shoveled charges was increased to equal that in the direct-filled ones, the reduced rate of percolation would much more than offset this advantage.

Under the above conditions the intermediate system only held its own, whereas, to justify its higher cost of installation and operation, it should show a decided superiority.

These results led to the rather unexpected conclusion that direct filling, when properly done, not only leaves a charge in as good condition for treatment as the intermediate method, but renders it possible to treat a larger proportion of the slimes. It may be suggested as an explanation that in a charge settled under water the grains of sand are free to arrange themselves most compactly, with a minimum of voids, and the slimes are held near where they originally settled, because the interstitial currents are not marked enough to disturb them; whereas, in charges shoveled into a vat with less than 15 per cent of moisture, the voids are a maximum, and the slimes which originally coated each grain of sand are washed off and settle through the charge, and, thus segregating, prevent uniform percolation. The fact that direct-filled charges never pack or settle more than 1 per cent, while indirect ones frequently contract over 10 per cent, during treatment, tends to confirm this view.

The final plant is located half-a-mile from the stamp-mill and 300 ft. vertically below it. The site, from which over 400 cords of timber were removed in clearing, is on a hill-side, and required about 5,000 yards of excavating and 630 yards of masonry for footings and retaining-walls. All the plant is housed, the buildings being heavily framed, as required by the deep snow-falls.

The vanner-tailings are carried to the plant by a box-launder 6 by 8 in. in section, and the excess of battery- and vanner-waters allows of a minimum grade of slightly under 5 per cent. At that grade, and with 1-in. riffles set 5 in. apart in the bottom of the box, the durability even of the local soft hemlock and cedar lumber used is quite satisfactory.

To make the required fall of 300 ft. to the plant, a series of drop-boxes is inserted where the topography is favorable; and a 12-in. sand-pocket at the bottom of each drop, to prevent wear, should also be an efficient saver of escaping amalgam or mercury.

The tailings enter the top of the storage-solution room; and when, as in the period under consideration, from 40 to 60 stamps are running, the classification is done in two boxes, the first being 18 by 24 in. wide by 18 in. deep, with two vertical sides and false ends sloping 60° to a 0.5- by 24-in. slot, under which hydraulic currents can be applied. This box settles

only the coarse sands, which are drawn off through a 2-in. nipple into the charging-laundry. The second box is pointed, 3 ft. 6 in. in width and depth, and 7 ft. long, with a 3-in. outlet. A plug-valve in this outlet, in combination with the hydraulic current in the box, regulates the speed of the distributors, while maintaining the same separating action in the box. The overflow from this box, carrying about 60 per cent of the water and 20 per cent of the total tailings, is run to waste. This overflow should all pass a 100-mesh screen, and is mostly impalpable material.

All of the vats and tanks, except the sumps, are of steel. The leachers, 32 ft. in diameter by 6 ft. deep, are on timber foundations with masonry footings. Caps and sills are parallel, an arrangement which renders jacking-up easier than the usual right-angled structure. Bents and posts are spaced 4 ft. between centers, and the bracing does not extend above the post, the shimming being done between posts and caps. On the caps are 4- by 8-in. joists, spaced 18 in. between centers. There is no flooring on any of the vat- or tank-joists, and there is head-room under them all, so that leakages occurring in the plant cannot be unknown or inaccessible. The vats are fitted with top annular-overflow launders; and, to preserve a level rim, a soft wood strip, projecting above the edge of the vat, is fastened with stove-bolts to the side-plates, the joint being calked with oakum. Any settling of the vats which is not so great as to require jacking-up from below can thus be easily rectified, just as with tongue-topped wooden vats.

The distributors are of the usual type, and have twenty 1.5-in. arms. This area of discharge-openings, with our quantity of tailings, never allowing any head to accumulate in the distributor-hopper, renders uniform distribution difficult. The slightest throw of the stream from the charging drop-box off the center of the hopper results in uneven charging, most of the slimes going to the low side of the vat. Many devices were tried to remedy this, and to prevent twisting of the drop-stream, which motion also has a classifying effect; and the last and simplest plan was successful. The drop was made square; and inside the lower end of it an iron strip was fastened to each side. The square drop prevents twisting, and to throw the stream toward the low side of the charge small wedges were driven under the iron strips, as needed.



These details may seem trifling; but when the object is to treat the maximum quantity of slimes by percolation they will be found of prime importance in obtaining a uniform charge.

Where there is enough head-room to give distributor-arms 3-in. grade per foot, arms of 1 in. diameter, or even smaller, could be used without danger of clogging. This would be an improvement upon the usual form, which has 1.5-in. arms at 1.5-in. grade; for it would approach the ideal conditions requiring a maximum number of arms fed from a full hopper.

Continuous, uniform charging is essential to make a good charge; hence, so far as possible, stamp-mill hang-ups were made between charges. When, however, an emergency hang-up was made with a charge incomplete, a gentle hydraulic current was started under the filter in the vat; the water-feed to the distributor was increased; and, when charging was recommenced, the settlers were run with a strong hydraulic current for a few minutes, allowing only the coarse sands to settle. Despite every precaution, however, a slime-layer will mark in the charge every mill hang-up of over ten minutes' duration.

The overflows from the vats and the separators were combined before they reached the overflow-sampler, so that I cannot give separate figures on the vat- and box-separations. The longest distributor-arms occasionally make a wave strong enough to throw over a little of the finer sands into the overflow-launders, so that of the combined overflow 1.1 per cent was caught on a 100-mesh screen. Much of this coarse residue, however, was wood fiber from the mine and flumes.

The classification of values in charges and overflows was marked, the average assays and screen-tests for the year being:

	Gold. Oz. Per Ton.	Silver. Oz. Per Ton.	Through 100-Mesh. Per Cent.
Vanner-tails, . . . . .	0.0838	1.008	64.86
Charges, . . . . .	0.0931	0.637	50.70
Overflows, . . . . .	0.0621	1.878	98.90

The high silver in the overflow is due to its association with galena in the slimes.

The system of classification, it must be admitted, was crude; and if the plant were pushed to its full capacity, close hydraulic classification, yielding a clean, quick-percolating charge, would be necessary; but, as run at this time, the system was very sat-

isfactory, for it settled and treated a total of 70.1 per cent of the vanner-tailings. When we were treating oxidized ore, charges could not be successfully treated that carried over 45 per cent of fines.

With all the stamps dropping, a vat was filled in from 28 to 32 hours; but during the last quarter of 1902 it required from 48 to 60 hours. The experimental work called for 5 days' treatment; but, with the reduced crushing, we increased the proportion of slimes settled, and treated charges as long as our capacity allowed,—the average time of treatment being 10.5 days. A normal charge was 185 tons, dry weight.

The acid tests seem, in the case of the Ymir tailings, to be worthless as an indicator for the use of lime. The ordinary tests for latent acidity called for over one pound of lime per ton. Phenolphthalein indicated more. We used 200 lb. to the charge for a while, until the zinc boxes grew so foul that dried precipitates assayed less than 1,200 oz. of fine bullion per ton. The methyl-orange indicator was more satisfactory. It called for less than 0.5 lb. of lime per ton. But the final practice, which gave the best results, was to use less than 0.1 lb. per ton, and apply it in 5-lb. lots, the last lot going on with the final dose of strong solution. The cyanide-consumption was thus reduced over 10 per cent; and the zinc-box crude precipitates passing through  $\frac{1}{8}$ -in. screen, rose in grade to over 5,800 oz. per ton. In treatment, the final filtrates became very cloudy when they titrated between 0.1 and 0.05 per cent KCN; and the small addition of lime, just before starting weak treatment, tended to keep the slimes coagulated and the filtrates clearer.

Vacuum-percolation was soon discarded, as it gave a very dirty filtrate and packed the charges, so that the final rate of percolation was slower, even with the vacuum, than it would have been if unassisted throughout.

The routine of treatment was as follows:

After leveling and adding dry lime, two 5-ton doses of weak solution (between 0.1 and 0.05 per cent KCN) are successively given. When these have disappeared they are followed by four or five 10-ton doses of strong solution (0.2 per cent) at 5-hour intervals. There is no trace of cyanide in the filtrate for 6 hours after the first charge of weak solution, and there are no values until after 15 hours. After adding 40 or 50 tons of strong solution, the charge is drained 6 hours, when the fil-

trate has almost ceased, and titrates from 0.04 to 0.07 per cent KCN. Twenty tons of strong solution are now run in under the filter. It requires from 6 to 8 hours to run this in without channeling the charge, and when it is all in, if drainage has been sufficient, there will not be more than 5 in. of solution on top of the charge. After soaking from 3 to 6 hours, the charge is drained, and a 10-ton top-treatment follows. Top- and bottom-treatments thus alternate until 160 tons of strong solution have been applied. Then follows weak treatment, applied on top in 5-ton doses as rapidly as it disappears, to a total of 40 or 50 tons. This is followed by wash-solution (0.05 per cent to trace of KCN), 40 tons; and this, by water as required, to preserve the balance of solutions or secure low end-filtrates. No solution is run to waste, other than what is lost as moisture in the residues.

The adoption of bottom-treatment had a marked effect. During the first 6 months' operating, when only top-treatment was used, the gold-extraction gradually fell off from 70 to less than 50 per cent; and assays from the bottom of the charges were discouraging. When the connections for bottom-treatment were made in October, the gold-extraction rose at once from 48 per cent in September to 75 for October; and the average actual gold-recovery for the last quarter of the year was 80.23 per cent, while the actual silver-recovery also rose from 25 to 41 per cent.

The total time of treatment was from 9.5 to 11.5 days. The average rate of percolation was 1 in. per hour, although 64 and 71 per cent gold-extraction were obtained from charges having only 0.5-in. and 0.6-in. rates. The highest rate during this period was 1.5 in. per hour. This slow percolation means high cyanide-consumption and long treatment; but it also means larger slimes-tonnage treated. It does not require a large increase in daily tonnage to pay for extra vats and a little more cyanide.

Residues are discharged through two 10-in. plug-flanges placed 4 ft. from the sides of the vats. A 2.5-in. hose with 2-in. nozzle is used under 115 lb. pressure, the hose being held in a swivel-clamp at the edge of the vat. The bottom 6 in. is cleaned out at reduced pressure. One man will sluice out a charge and clean the filter in from 3.5 to 4 hours.

The filter is made of three thicknesses of 16-oz. burlap, hav-

ing the usual false bottom and rope-grouting, and anchored to the vat-bottom to counteract its buoyancy.

Precipitation is effected by zinc-shavings in eight 10-compartment steel boxes, and it is perfect with the weakest solutions when run at the rate of one ton of solution per 24 hours per cubic foot of shavings. The shavings in the wash solution-box are lead-dipped. Whenever there were values in the effluent, running at the normal rate, they were found to be due to precipitate mechanically carried over after dressing boxes, and it was found best not to disturb a box after it had been running more than a week, but to add shavings on top as required.

The clean-up method, while it just about competed with refiners' rates, was not satisfactory. The precipitate was dried, fluxed and smelted direct, without either acid or roasting-treatment and without refining, and produced a bullion of 720 total fineness. The treatment-loss was 1 per cent of the gold and 0.5 per cent of the silver. The clean-up cost, including freight and treatment of the resulting high-grade zinc-slag, was 17 cents per pound of dried slimes. This does not include charges for bullion shipment.

Operating-costs per ton in detail for the last quarter of 1902, when 7,150 tons were treated, were as follows:

*Cost of Treatment per Ton.*

Cyanide, at \$0.22 per lb.,	.	.	.	.	.	.	.	\$0.200
Zinc, " 0.09 "	.	.	.	.	.	.	.	0.020
Lime, " 0.009 "	.	.	.	.	.	.	.	0.001
Fuel, " 2.50 per cord,	.	.	.	.	.	.	.	0.019
Clean-up supplies,	.	.	.	.	.	.	.	0.029
Repairs,	.	.	.	.	.	.	.	0.014
Miscellaneous,	.	.	.	.	.	.	.	0.003
Labor (wages at \$3.50 and \$4.50 per day),	.	.	.	.	.	.	.	0.234
Total,	.	.	.	.	.	.	.	<u>\$0.520</u>

Assay-office costs were not allocated; but the cyanide-plant proportion should approximate \$0.025 per ton. These results were obtained when treating but 78 tons per day. This cost can be reduced to 38.6 cents when the plant is treating its full-rated capacity of 200 tons per day; and the introduction of reverberatory and cupel furnaces for clean-up would still further reduce it.

The plant is equipped with steam-heating and electric-lighting plants, and has an auxiliary water-power system capable of

developing 300 H.P. The pumps and lighting-dynamo are run by Pelton water-motors under 320-ft. head; and the only cost of power is the trifling item of maintenance of flumes.

The total cost of the works, including the power-plant, was \$57,951.63. Omitting the costs of the main mill-building, boarding-houses, power-plant, and heating-system, and thus reducing the estimate to the basis of an open-air plant, the cost was \$33,782, including the clearing, excavating and masonry, previously referred to, which amounted to \$6,321.34.

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### Hearth-Area and the Number of Tuyeres in Iron Blast-Furnace Practice.

BY F. L. GRAMMER, BALTIMORE, MD.

(New York Meeting, October, 1903.)

THE rapid advances on the mechanical side of blast-furnace practice have not always been accompanied by a proper appreciation of the metallurgical requirements. This is illustrated occasionally by the improper use of automatic barrow charging-devices and bins (introduced at the expense of accuracy in weight), with ores of irregular composition or ores that have been poorly graded. The 100-ft. blast-furnace has not always been advantageous, and in many instances the increase in the number of tuyeres, from 8 or 12 to 20, has resulted in decreased output. The data pertaining to this division of blast-furnace practice are now sufficiently complete to allow inferences to be made.

The use of fine ores has indicated the manner in which large outputs could be obtained, as it has emphasized the fact that intimacy of contact between gases and ore, and coke and air, were more important than the time of exposure. It was reasonable to suppose that if more points of contact were made between the air and the coke, their union would occur in a shorter period of time, and the coke would be burned more rapidly in the same-sized hearth, and an increased output thereby obtained. In consequence of this theory, many furnace men upon relining a furnace increased the number of tuyeres from 8 or 12 to 20, or even more, frequently with unfortunate results. If the number of tuyeres in an old furnace is

increased while it is in blast and banked, there will be quite an increase in the output on resuming operations for a day or two, owing to walls having been cleaned. A single advantage which was obtained in one or two cases by the use of multiple tuyeres has been offset by many disasters and, in at least one instance, the furnace wall was cut out within a period of two weeks. Unless the penetration of the blast is properly calculated, there is danger that it will go through the center or up the walls of the furnace.

Apart from the increase in the number of tuyeres, the points of horizontal contact also were increased by "oval" tuyeres; "three-way tuyeres;" and a very ingenious "fan-spraying nozzle," which has been described in the *Iron Age*, November 25, 1897.

These patented tuyeres, while hardly accomplishing the object for which they were designed, are not without merit. I have used the three-way tuyere very advantageously on furnaces having a dirty crucible, or having a lodgment on the wall, and by its use I have obtained a clean hearth. Owing to certain objections, however, these devices have never been extensively adopted in regular furnace practice, for if increasing the number of tuyeres had shown more benefits, the practice would have been more widely adopted. The metallurgists of the present generation have experimented with the number of tuyeres used, but no one, so far as I know, has had the temerity to place them in more than one horizontal plane in the last ten years.

The number of tuyeres required in an iron blast-furnace must be determined from the hearth-diameter, the fuel and ore used, and the grade of metal desired.

Consider first the subject of blast-pressure. In a furnace having an 8-ft. hearth, the average distance through which the air at one point has to travel, in order to cover the whole hearth-area, is the distance from the nozzle to the center of the area, or 4 ft. If the air is introduced at 2 points, the distance is  $0.5756 R = 2.3024$  ft.; at 4 points,  $0.4 R = 1.6$  ft.; at 8 points,  $0.3482 R = 1.3928$  ft.; at 16 points,  $0.3376 R = 1.3504$  ft.; and at an indefinite number (continuous),  $0.3333 R = 1.3333$  ft.

As far as assisting blast-pressure is concerned, with a hearth of the same diameter, much relief in reducing the pressure cannot be expected beyond the use of 8 tuyeres. Indeed, the pressure necessary to overcome the resistance offered, as represented by the distance from the circumference to the center of the area,

will be greater in proportion to the number of tuyeres, because of the greater friction offered by the smaller and more numerous channels which convey the air through the crucible. The 3 per cent assistance rendered by decreasing the distance from 0.3482 R to 0.3376 R is more than offset by the increased friction. The usual experience in using coke and a moderate quantity of fine ore is that, for the same volume of air and the same hearth-area, the blast-pressure with 16 tuyeres will be from 1 to 2 lb. higher than with 8 tuyeres.

Using from 16 to 20 tuyeres usually means that fewer tuyeres are lost at scattered intervals by the splash or the drilling of molten iron; the tuyeres wear out rather than burn out, and when one is destroyed the others follow quickly. With a larger number of tuyeres the settling of the charge is apt to be more even than with fewer tuyeres. Also, the dripping of the iron is less marked at any one tuyere. An observation through the peep-sight in the tuyere-stock shows that the activity of the movement of the coke is noticeably less with 20 tuyeres than with 10; the volume of the blast being the same in both cases and the diameter of the tuyeres being such as to give the same penetration of the blast. However, a few extra tuyeres are of advantage when the furnace is working irregularly; as it is very easy to plug the extra tuyeres with clay when not needed. Appreciating the fact that it is safer to have extra tuyeres, the tendency—even where no dreams of great benefit exist—is to increase their number.

For a given hearth-area the number of tuyeres that should be used is best determined by a comparison with the workings of a furnace of similar dimensions and engine equipment, operating under similar conditions of ore, flux and fuel, and grade of metal produced.

Excellent results have been obtained with Lake Superior ores in an 85-ft. furnace having a 11 or 12-ft. hearth and 8 tuyeres. For the purpose of illustration, assume a 11-ft. hearth, having the tuyeres uniformly spaced; in this case each tuyere of

equal diameter will cover an area of  $\frac{\pi D^2}{4} = 11.87$  sq. ft., or,

for all practical purposes, 12 sq. ft. If it was decided that the engines and stoves permitted a 14-ft. hearth,

$$\frac{\pi D^2}{4} = 13 \text{ tuyeres.}$$

11.87

would be the proper number to have, and on account of columns, it is probable that 12 or 14 would be the number selected, the even number being more easily placed in the construction.

If with a 11-ft. hearth and 8 tuyeres, a blast-pressure of 10 lb. per sq. in. were produced, the pressure calculated for 14 tuyeres and a 14-ft. hearth would be as shown in the following formula:—

$$\frac{7}{5.5} \times \left( \frac{3482}{3400} \right)^2 \times 10 = 13.2 \text{ lb.}$$

and adding 1 lb. for increased friction, the final blast-pressure would be about 14.25 lb. Based on this reasoning, an increase in the number of tuyeres will be beneficial.

In the light of our present knowledge it is assumed to be desirable that the horizontal area exposed to a single tuyere, or to a unit volume of air, should be constant for the same stock used.

Immediately after tapping the iron, the coke from 6 to 12 ft. below the tuyeres is exposed for combustion. As the level of the molten slag and metal rises the coke becomes covered, and the air must seek coke at greater and greater vertical height above the tuyeres, until at the flushing-time the entire combustion, which has now become decreased in intensity, is above the tuyere-level. After flushing, the lower limit of the combustion-zone drops to the level of the cinder-notch, which is generally 2 ft. below the tuyeres. It is fair to assume that, at the same temperature of blast and with the coke of the same size and porosity, a unit volume of blast will attack a given volume of coke. Therefore, if the horizontal influence of a tuyere be restricted, and the blast-volume remain constant, the vertical influence of the blast will be increased.

A hearth of small diameter means that the combustion-zone extends to a greater height in the furnace than if the hearth were wide and receiving the same volume of wind. It means, also, that the heat is somewhat more intense, as is shown later in this paper. This explains in some measure the fact that the silicon-content of pig-iron is usually higher with furnaces having a narrow crucible than with those having a wide crucible. In the former the dripping slag and metal remain longer in contact with the incandescent coke before reaching the bottom of the crucible, and a part of the coke is at a higher temperature than is usual in a furnace having a wider hearth.



I do not say that the consumption of fuel is better in furnaces equipped with a small crucible; because there are too many influences which act the other way. With a small crucible, more heat is lost per ton of metal produced, on account of the heat conveyed away through the cooling devices; and in consequence there is less liability for the metal to break out. But, generally, small crucibles assist the assimilation of silicon, and furthermore, under ordinary conditions they can receive a hotter blast without sticking up the furnace than can the usual modern "jumbo." On the other hand, the increase of the horizontal hearth-area allotted to a unit volume of the blast should decrease the vertical height of the combustion-zone. If this be correct, a 70-ft. furnace would derive more benefit by increasing the number of tuyeres than would a 100-ft. furnace, because, by decreasing the height through which combustion takes place, we have practically added to the height of the furnace, and for Lake Superior ores a 70-ft. furnace is probably too low for best economy of fuel. An increase in the number of tuyeres is analogous to an increase in the area of horizontal influence.

My assumption, that each point in the horizontal plane of the hearth at the level of the tuyeres receives the same quantity of air, is probably exaggerated, even with coke. With a sticky fuel or with fine ores poorly distributed, it is quite certain that the air does not pass around the fuel as freely as water would fill the interstitial space in a pail filled with marbles. Under these conditions, the more contact possible between air and coke the better. Two reliable instances of marked improvement, resulting from an increase in the number of tuyeres without increasing engine-power or hearth-diameter, were obtained in a low furnace treating a charge of 75 per cent Mesabi ore, and in a furnace using anthracite fuel.

If small hearths assist to make high-silicon iron, large hearths tend to make low-silicon iron; in other words, an increase of the number of tuyeres tends to lower the silicon in the pig-iron produced in the furnace.

With 8 tuyeres, the tuyere is 0.3482 ft. distant from the center of its area of control, which extends over the entire space in front of the tuyere. In a hearth 12 ft. in diameter, the deflection that air would have to one side, before it meets air from the adjacent tuyere, is shown in the following equation:

$$\frac{\pi 12 (1 - 0.3482)}{16} = 1.53 \text{ ft.} = 18.4 \text{ in.},$$

or, with 16 tuyeres,

$$\frac{\pi 12 (1 - 0.3376)}{32} = 0.78 \text{ ft.} = 9.4 \text{ in.}$$

Using a hard coke, no marked improvement or change could be expected by decreasing the wall of coke below 18 in. to 9 in. Coke in a finely divided form, and anthracite of small sizes, would hardly find a thickness of 18 in. a barrier to the passage of air. This may explain why many have noticed no improvement in furnaces working under regular and healthy conditions of large and strong coke by increasing the number of tuyeres on the same hearth-area.

In determining the size of hearth, it is obvious that the character of fuel must be taken into consideration. A fuel with cellular structure, aside from its disadvantages in the upper part of the furnace, will burn more rapidly than one more dense. Coke having a cellular space of 45 cu. in. (a unit size), compared with charcoal having 150 cu. in. cellular space in same volume, certainly requires a different dimension for the hearth-area. Of course, coke is not quite so far behind charcoal as the ratio of 45 to 150, because the combustion of coke is assisted by the great pressure under which these furnaces are operated. The hearth-area, besides being controlled by the fuel, should increase in proportion to the additional volume of air deliverable by the engines, as well as to the heat obtainable from the stoves. It required nearly a decade to learn that an increase of 40 per cent in the diameter of the hearth necessitated a 100-per cent increase in the engine-equipment of the plant, not to mention the stove-capacity.

The other features of the hearth, such as height of cinder-notch and iron-notch, cubic capacity compared to total furnace-capacity, etc., have been discussed in former contributions to the *Transactions*.

We have discussed the pressure, diameter of hearth and number of tuyeres, for metal of the same grade using a constant and an increased blowing-capacity; yet the selection of the most desirable diameter of hearth for a set of entirely new conditions is a problem which can only be settled empirically.

The present treatment of Lake Superior ores with Connells-ville or hard coke in furnaces equipped with modern engines is to have a volume of air sufficiently large to fill the empty crucible once every 2 seconds. A unit volume of air attacks a unit volume of coke, presenting a constant area of surface; and it may be said further that, at the same temperature, the quantity of coke burned varies almost directly as the weight of air supplied. If the volume of air per pound of fuel consumed did not vary from 53 cu. ft., under the best conditions, to 120 cu. ft., in plants having leaky valves, poor engines and split stove casings, we could speak of volumes in place of weights.

In the year 1883 Mr. Witherow thought that at a temperature of from 1,300 to 1,500° C. 140 cu. ft. of air per minute, in a furnace using coke or anthracite, would produce 1 ton of cast-iron in 24 hours; and using charcoal, 110 cu. ft. of air per minute would produce 1 ton of cast-iron. At the present time 85 cu. ft. of air will produce 1 ton of cast-iron under conditions similar to those just quoted. This difference is really very slight, considering the lower fuel-consumption incident to the lower silicon-content of the resultant cast-iron, together with the richer and more easily reduced iron-ores used.

It is known, also, that at higher temperatures iron dissolves more carbon, *i.e.*, makes richer iron, and, other conditions being equal, the iron will contain more silicon. Thus, in order to increase the percentage of carbon and silicon in the cast-iron produced, the temperature of the furnace must be increased, which is accomplished by using less ore and a more siliceous slag.

It is known that the reduction of magnetites requires a high temperature of the gases escaping from the furnace, or that the gases should be rich in carbon monoxide and, consequently, the quantity of coke burned for a given unit of iron is greater than that resulting from the treatment of Bessemer ores. These remarks apply also to ferromanganese and to silico-spiegel in a greater degree; as the cooling effect of the descending stock does not reach so low in the furnace and, consequently, the temperature in the crucible is higher. The use of soft fuel which will dissolve in the top of the furnace means that more fuel will be required to do the work in the crucible.

A porous fuel, like charcoal, with the pores open and not

sealed, as in coke, offers from 8 to 10 times the contact surface in a given weight of fuel for attack by the blast, and its combustion therefore occurs lower in the furnace. The occluding power of charcoal causes it to ignite at a very low temperature. If it were desired to make an iron at a heat near to its freezing-point, such as basic-iron or forge-iron, a fuel easily ignited (charcoal) would be advantageous, especially if cheap and abundant.

Schinz, in 1868, says that the one point about furnace-lines of greatest unanimity of opinion is, that wide hearths are better adapted to forge-iron and narrow hearths to foundry-iron. He also gives the calorific intensity of coke about  $100^{\circ}$  higher than that of charcoal.

The better sorting of the ores, the more uniform quality of the coke and the more regular charging of the furnace have resulted in reducing the irregularity in the working and have diminished the demand on the heat of the crucible, consequently a smaller margin of heat in the crucible is permissible and the silicon-content of the resultant cast-iron becomes lower. The crushing of the ores and the lowering of the bosh has allowed the silicon-content of the cast-iron to be lowered without danger of increasing the sulphur-content, or of having the charge stick on account of the larger ratio of lime to total silica-content in the burden. These results have met with the approval of those interested in developing the practice of smelting Bessemer ores.

A wide hearth will contain a large quantity of molten metal and, acting like a "mixer," it reduces any temporary irregularity in the composition of cast-iron, thereby yielding a product of greater uniformity. A soft fuel, with a dull lustre, like charcoal, radiates and absorbs heat more rapidly than a hard fuel, with metallic lustre, like coke or anthracite.

In less than 30 years the ratio of hearth-area to greatest bosh-area has increased from  $\frac{1}{16}$  to  $\frac{9}{16}$ , and the furnace-capacity in cubic feet per ton of metal produced has changed as shown in the table on page 616.

The data given in the table show that the fine size of the ore allows a lower silicon-content of the pig-iron produced and yields a better contact of blast, which permits an increased volume of air to be used and a corresponding decrease in the time of exposure.

*Change in the Factors of Iron Blast-Furnace Practice.*

Time.	Place.	Cu. Ft. of Furnace Per Ton of Metal Produced in 24 Hours.	Remarks.
1854.....	England.	190	{ Resulted from increasing the height of furnace from origi- nal 45 ft. to 75 ft. and more. Height of furnace from 65 to 75 ft.
1870.....	England.	380	
1876.....	United States.	197	
1889.....	United States.	59	{ Lake Superior ores; smaller proportion of lump sizes; crushed more thoroughly; lower bosh; large volume of air.
1891.....	United States.	50	{ Mesabi ores; wider hearth; in- creased volume of air; lower silicon in iron.
1903.....	United States.	20 to 30	

Schinz says that the time of exposure in the reduction zone should be relatively 4.53 for spiegeleisen, 2.75 for foundry-iron and 1 for forge-iron. He says, also, that these ratios can be reduced by increasing the volume of air.

Gruner gives, as a desirable diameter for the crucible of furnaces using charcoal, 20 in. up to 9 ft., and for furnaces using coke, up to 10 ft. For furnaces using coke he gives the following cubic capacity of the crucible: Furnaces producing from 2 to 5 tons of pig-iron in 24 hours, from 18 to 36 cu. ft.; producing from 5 to 10 tons in 24 hours, from 36 to 62 cu. ft.; producing from 10 to 20 tons in 24 hours, from 72 to 144 cu. ft.; and producing from 40 to 60 tons in 24 hours, from 216 to 288 cu. ft.

Modern American practice is from 3 to 3.5 cu. ft. of crucible-capacity per ton of metal produced in 24 hours. Gruner says that for lean ores the size of hearth is greater per ton of metal produced than for rich ores. These early authorities are quoted merely to show their ideas on the subject.

Practically, for the same volume of air the silicon-content of the resultant cast-iron may be raised, or a more refractory charge reduced, by lightening the burden, an act which raises the zone of incandescent coke and usually increases its temperature. The temperature of the hearth also may be increased by raising the temperature of the blast. On the other hand, on those ores and other materials which have not been proven to

be as independent of time-exposure as are Lake Superior ores, the inclination is to use a lesser quantity of blast, which suggests a smaller hearth-diameter commensurate with the reduction of blast, in order to give a longer time of exposure.

For porous materials like charcoal, conditions opposite to those for coke hold, and being soft it requires a smaller hearth, because of the pressure of the superincumbent charge and because it offers so large an area of surface for attack.

For hard, dense materials like coke, offering fewer surfaces of attack, with a metal requiring small fuel margins, such as basic-iron, a shape of furnace is desirable which will allow the quick removal by the rising gases of the heat generated, and that shape is a wide hearth. Furthermore, a wide hearth offers the proper volume of coke to the attack of the blast, and with a low vertical ignition the spreading of the heat is facilitated laterally more than vertically for the same quantity of air. This may mean a slightly lower temperature at the hottest region.

I hesitate to say exactly what is the proper hearth for the production of an iron of a given silicon-content, because economy in coke consumption is regarded differently in different localities. Some metallurgists think that where coke is expensive, the hearth-area in making foundry-iron should bear approximately the same relation to the hearth-area in making Bessemer metal as the Bessemer time-exposure does to the foundry-iron time-exposure. This is to be the relation if the furnaces producing foundry- and Bessemer-irons have the same stove- and engine-equipment, the same richness of ore and the same nature of the ore apart from phosphorus. This statement is crude, but it holds true in general. The natural field for experimenting on the dimensions of the blast-furnace has been in the furnaces producing Bessemer-irons, as they are usually assembled in larger units near together, and hence are more easily compared and studied.

"Inversely as the coke consumption" might be considered a better rule for varying the diameter of the hearth.

When the relations between hardness and calorific intensity, porosity and rapidity of ignition, softness and solubility in carbon dioxide—disregarding chemical percentages—have been shown by our technical schools, we can speak other than by rule of thumb.

## The Geology and Copper-Deposits of Bisbee, Arizona.\*

BY F. L. RANSOME, WASHINGTON, D. C.

(Albany Meeting, February, 1903.)

THE following paper aims to present in a much condensed form the salient results of a detailed geological study of the Bisbee quadrangle, Arizona, carried on during the latter part of the year 1902. It is essentially an abstract of the final official report which will shortly appear in the series of professional papers of the U. S. Geological Survey, accompanied by maps, sections and illustrations.

### I. GEOGRAPHY.

The Bisbee quadrangle, which includes the Warren mining district, lies in Cochise county in the southeastern part of Arizona. It is inclosed between the meridians  $109^{\circ} 45'$  and  $110^{\circ} 00'$  and the parallels  $31^{\circ} 30'$  and  $31^{\circ} 20'$ , the latter being locally the Mexican boundary line. The area of the quadrangle is about 170 square miles, and includes the southeastern half of the Mule Mountains, one of the smaller of the isolated ranges characteristic of this part of Arizona. The Mule Mountains, while less markedly linear than the Dragoon, Huachuca, Chiricahua, and other neighboring ranges, have a general NW-SE. trend. They may be considered as extending from the old mining town of Tombstone to the Mexican border, a distance of about 30 miles. On the northwest they are separated by the broad, flat floor of Sulphur Spring Valley from the Chiricahua Range, and on the southeast by the similar broad valley of the Rio San Pedro from the Huachuca Range. The maximum width of the range is about 12 miles, and its greatest elevation attained by Mt. Ballard, west of Bisbee, is 7,400 ft. above sea-level. An index-map, showing the geographical position of the Bisbee quadrangle, is shown in Fig. 1.

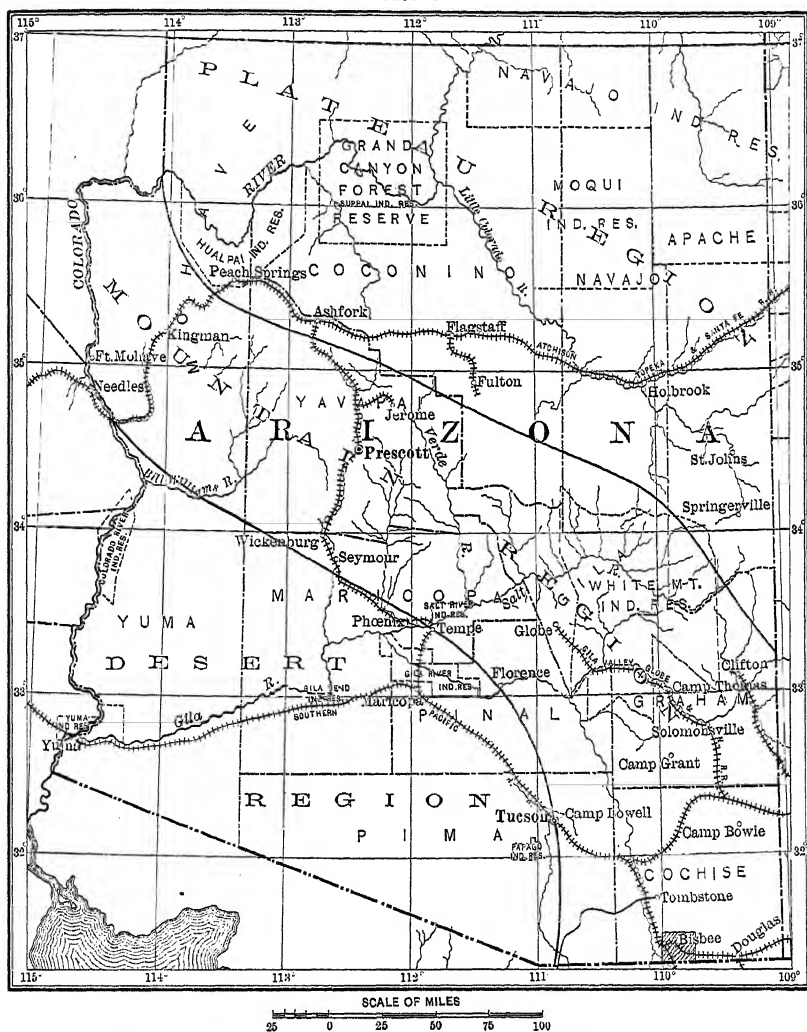
The town of Bisbee (Fig. 2), with an estimated population of 6,000, is crowded into a few narrow confluent ravines near the

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heart of the range, 7.5 miles north of the International boundary. It is connected by the El Paso and Southwestern Railroad to the east with the new town of Douglas in Sulphur Spring Valley, with Deming and with El Paso. To the west, branches

FIG. 1.



INDEX-MAP, SHOWING THE POSITION OF THE BISBEE QUADRANGLE, ARIZONA.

of the same road run to Naco on the Mexican boundary, to Tombstone, and to Benson on the main line of the Southern Pacific Railroad. At Douglas, connection is made with the Nacosari Railroad to Cos, and at Naco with the Cananea, Yaqui



River and Pacific Railroad to Cananea, thus bringing the new smelters at Douglas within reach of the important mines at Nacosari and Cananea in the State of Sonora, Mexico.

## II. GENERAL GEOLOGY.

The first attempt to describe the geological environment of the Bisbee copper-ores was that of Wendt,<sup>1</sup> in 1886, followed 13 years later by the more comprehensive account by Douglas.<sup>2</sup> Both these papers, however, touch but briefly on the geology and structure of the Mule Mountains, being concerned mainly with the ore-bodies and the metallurgy of the ores. Dumble's<sup>3</sup> reconnaissance notes published in 1902, although necessarily fragmentary, are really the first step toward any general knowledge of the broader geological features, and to him is due the first recorded recognition of the extensive Cretaceous deposits of the Bisbee region.

The rocks of the Mule Mountains may be primarily divided into five main groups: (1) a pre-Cambrian basement of crystalline schists; (2) a Paleozoic sedimentary series; (3) a Mesozoic sedimentary series; (4) Quaternary valley-deposits; and (5) Intrusive igneous rocks.

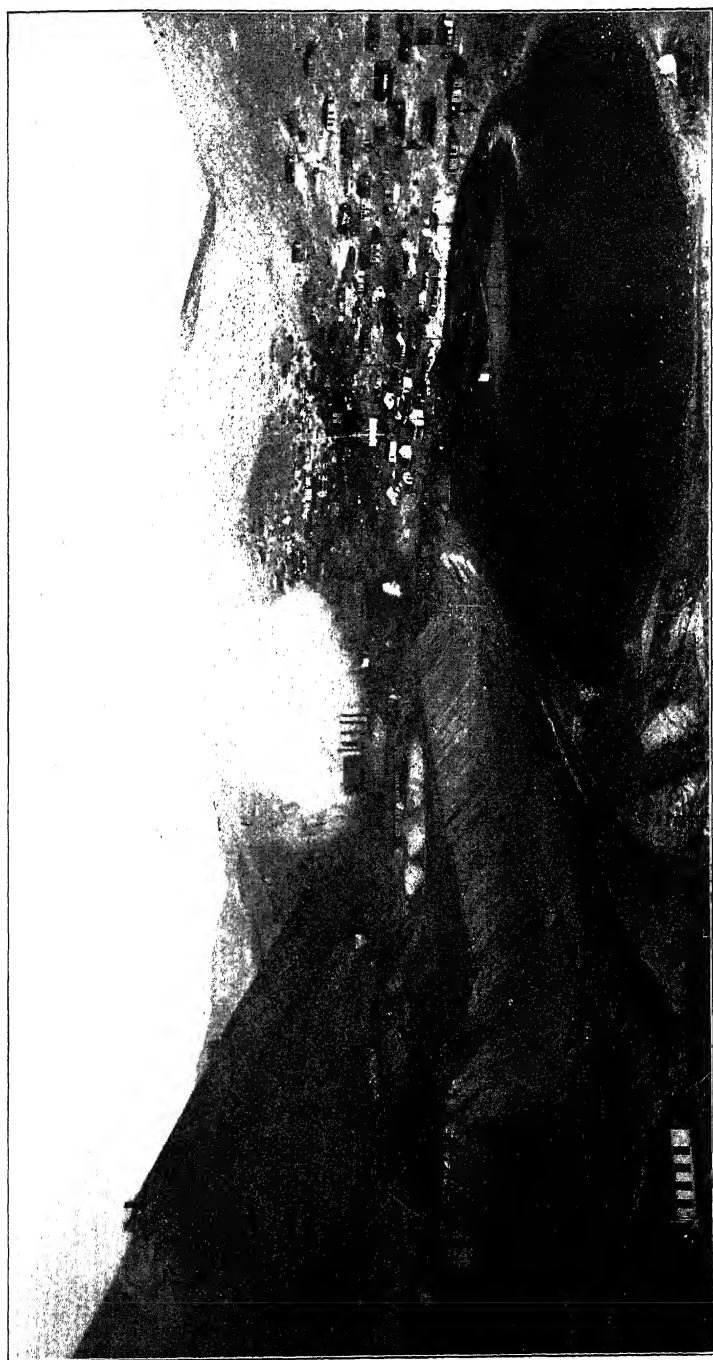
One, standing upon Mt. Ballard, can look out over the clustered ridges and peaks composing the mountain range and to a certain extent trace the general distribution of the rocks by the character of the topography. The Quaternary deposits floor the broad valley-plains stretching away from the intricately embayed line marking the foot of the mountains. The NW-SE. diagonal of the Bisbee quadrangle, a line which passes very near the town of Bisbee, and corresponds with the course of Tombstone Canyon, roughly divides the mountainous part of the quadrangle into two topographical divisions. Northeast of this line the mountains are sculptured from comparatively soft Mesozoic beds, striking approximately with the trend of the range (NW-SE.) and dipping at generally moderate angles toward Sulphur Spring Valley on the northeast. The slopes of these hills are comparatively smooth, although

<sup>1</sup> "The Copper-Ores of the Southwest," *Trans.*, vol. xv., pp. 25-77.

<sup>2</sup> "The Copper Queen Mine," *Trans.*, vol. xxix., pp. 511-546.

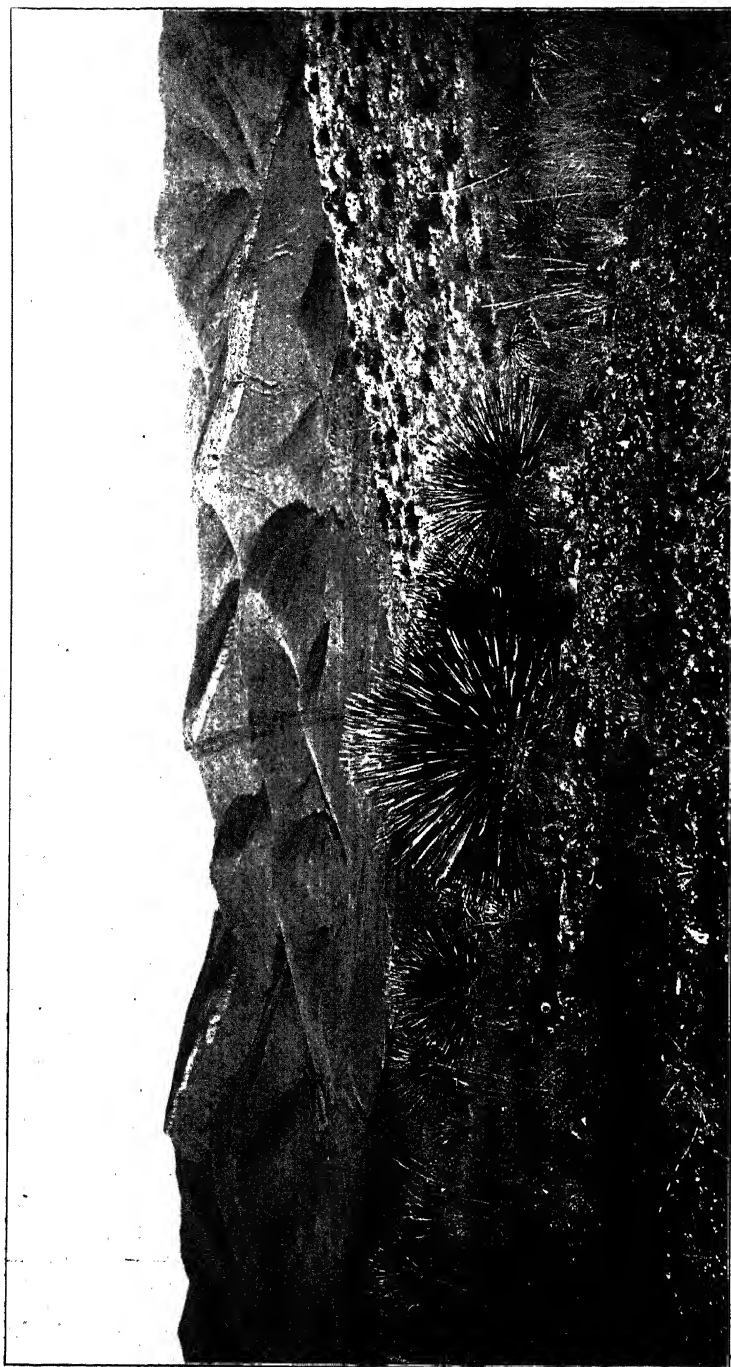
<sup>3</sup> "Notes on the Geology of Southeastern Arizona," *Trans.*, vol. xxxi., pp. 696-715.

FIG. 2.



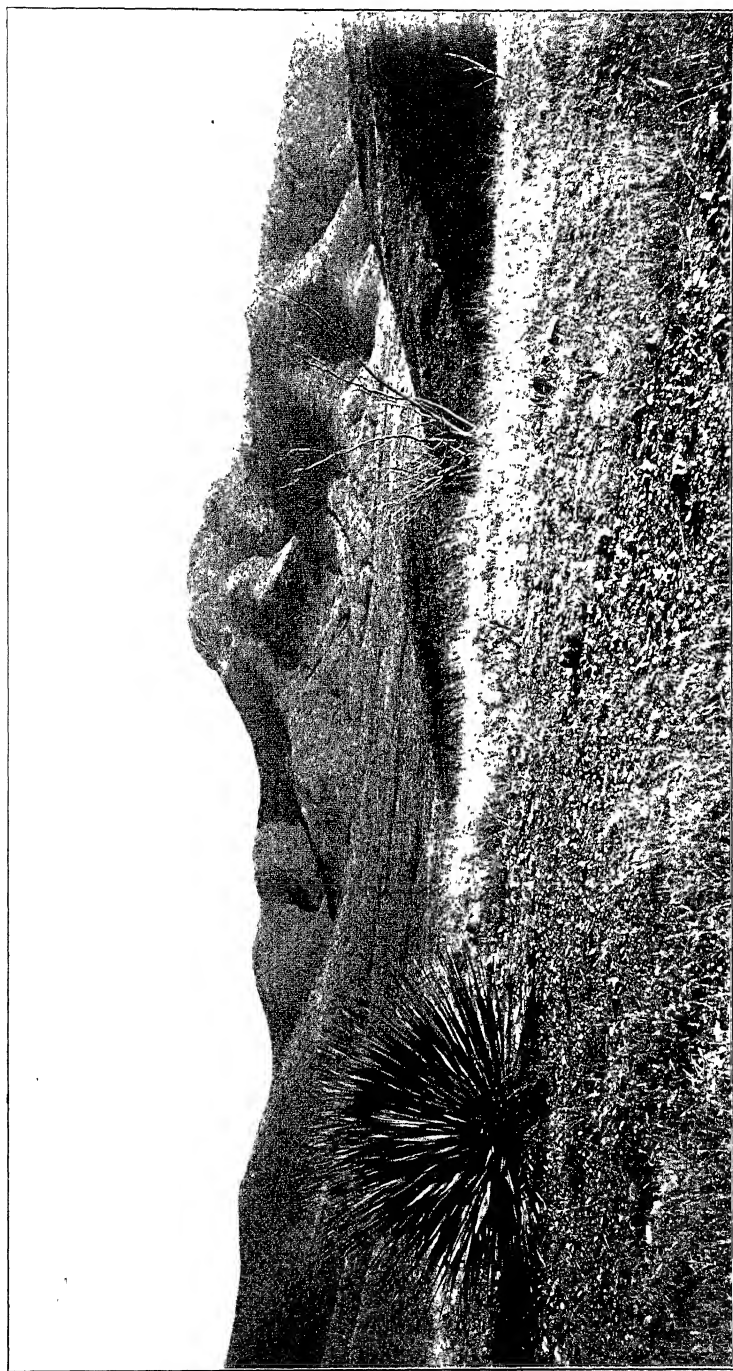
THE TOWN OF BISBEE, LOOKING NORTH FROM SACRAMENTO HILL.  
The Copper Queen Smelter and the Czar Shaft of the Copper Queen Consolidated Mining Company are Shown in the Middle of the Picture.

FIG. 3.



View of the Cretaceous Beds of the Bisbee Group, Southeast of Bisbee, Looking North across Mule Gulch.

FIG. 4.



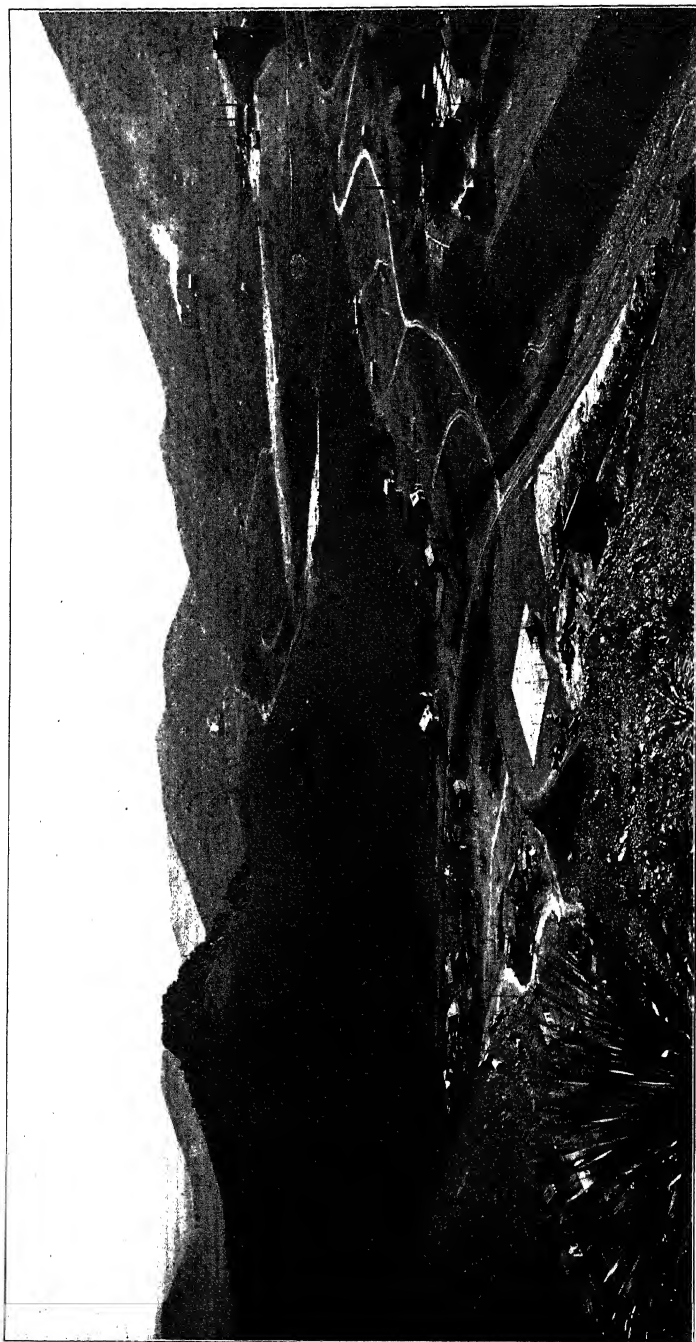
Gold Hill, Viewed from the North, Showing the Paleozoic Limestones Thrust from the Southwest over the Cretaceous Beds.

Spray Mine.

Holbrook Mine.

FIG. 5.  
Lowell Mine.

Calumet & Arizona Mine.



Sacramento Hill, Viewed from the North, Showing the Semi-Circular Distribution of the Principal Mines in the Limestones about the Western Half of the Porphyry Stock. The Dividend Fault passes under the Slag Dump in the Foreground of the Picture.

the occurrence of a hard fossiliferous limestone in the middle of the generally arenaceous group has occasioned a persistent cliff of erosion, conspicuous from points near Bisbee. A view of the Cretaceous beds of the Bisbee group, southeast of Bisbee, looking north across Mule Gulch, is shown in Fig. 3. Southwest of this line, pre-Cambrian schists and Paleozoic beds, both cut by eruptive rocks, prevail. These are more resistant of erosion and more heterogeneous in character than the Mesozoic beds and have a far more complicated structure. Their degradation has consequently resulted in a more rugged and less regular topography.

#### *The Pre-Cambrian Basement.*

The rocks forming the pre-Cambrian basement, the oldest rocks known in the Mule Mountains, are fine-grained sericite schists, derived from ancient sediments. These, which have been called the Pinal schists, were probably originally shales or arkose sandstones, which were folded and metamorphosed into their present crystalline condition before Cambrian time.

#### *The Paleozoic Sedimentary Series.*

After long erosion the Pinal schists were reduced to a surface of very slight relief, which in Cambrian time was submerged beneath the sea and covered with the sands that are now represented by the Bolsa quartzite (Cambrian), from 400 to 500 ft. in thickness. The Bolsa quartzite has at its base a thin conglomerate, which rests with profound unconformity upon the eroded edges of the Pinal schists.

The submergence of the area continued, and about 750 ft. of thin-bedded, cherty, fossiliferous Middle Cambrian limestones, named the Abrigo limestone, accumulated on top of the Bolsa quartzite. No record of Silurian time has been discovered in the Bisbee quadrangle. Overlying the Abrigo limestone, apparently in perfect conformity, are 340 ft. of dark-colored, compact, rather thin-bedded limestones, with some intercalated shales, all carrying an abundant and characteristic Devonian fauna, consisting chiefly of brachiopods and corals. This formation is distinguished as the Martin limestone.

The opening of Carboniferous time was, in this region, unmarked by any interruption of the continued subsidence. No

unconformity has been detected between the Devonian and the lower Carboniferous (Mississippian) rocks. The latter consist of white or light-gray granular limestones, often made up almost entirely of crinoid stems and containing a fairly abundant brachiopod and coral fauna. The thickness of the lower Carboniferous limestone may be provisionally given as 700 ft. The beds are often 6 ft. or more in thickness, and commonly form cliffs overlooking slopes carved from the less resistant Devonian and Cambrian limestones. This formation has received the name of the Escabrosa limestone.

There is in the Mule Mountains no discoverable stratigraphic break between the lower and upper Carboniferous beds. Subsidence apparently continued, and the generally thinner beds of upper Carboniferous (Pennsylvanian) limestone accumulated to a thickness of over 3,000 ft. above the lower Carboniferous. The upper Carboniferous limestones are usually more compact in texture than those of the lower Carboniferous, and are more fossiliferous. They are also somewhat more variable in color, pinkish and yellowish beds being of frequent occurrence. This formation has been named the Naco limestone.

The local Paleozoic section from the Pinal schists very nearly to the top of the Escabrosa limestone is well exposed on the northeast face of Escabrosa Ridge, about 1.5 miles west of Bisbee. The beds of the Naco limestone are best seen in the hills just north of Naco Junction (5 miles southwest of Bisbee), and the conformable relation between the Escabrosa and Naco limestones is well shown near the Whitetail mine, about 2 miles due south of Bisbee.

#### *Post-Carboniferous Faults and Intrusions.*

At some time during the interval between the close of the Carboniferous and the opening of the Cretaceous, the long-continued subsidence and sedimentation of the region were interrupted by extensive faulting, probably connected with uplift. The faults were mostly of the normal type, but there was also some reversed or overthrust faulting that may have been slightly later than the prevalent normal faulting. Following the faulting came intrusions of granitic magma, which solidified as granite, granite-porphyry and rhyolite. These intrusions took the form of dikes following fault fissures, of sills injected be-

tween sedimentary beds, and of irregular stock-like masses. The dikes are well shown along the southwest face of Escabrosa Ridge west of Bisbee. The larger intrusions are exemplified by the granitic mass of Juniper Flat, which is enclosed in Pinal schists, and by the smaller body of mineralized and altered porphyry forming Sacramento Hill, just southeast of Bisbee, and intrusive into Pinal schists and Paleozoic limestones. The latter mass is of particular significance from its connection with the principal copper-deposits of the district. The intrusion of the porphyry of Sacramento Hill was accompanied and followed by contact metamorphism involving the introduction of much disseminated iron pyrite into the adjacent Paleozoic limestones and the local transformation of the latter into fine-grained aggregates of calcite, tremolite, wollastonite, garnet, chlorite and quartz. The metamorphic minerals, with the exception of the pyrite, are never conspicuously developed, and can usually be identified only in microscopic sections.

After the intrusion of the granite-porphry, the region was eroded until the opening of Cretaceous time. It is probable that the primary mineralization of the district followed closely upon the eruption of the porphyry, and thus dates from early Mesozoic time.

#### *The Mesozoic Sedimentary Series.*

At the beginning of the Cretaceous the region again began to subside, and a conglomerate was deposited by the advancing sea over the eroded surface of the pre-Cambrian and Paleozoic rocks, with their intruded masses of porphyry. This conglomerate is the basal member of the Bisbee group, an assemblage of Cretaceous strata comprising four formations. These are in ascending order (1) the Glance conglomerate, (2) the Morita formation, (3) the Mural limestone, and (4) the Cintura formation.

In places the Glance conglomerate was laid down to a uniform thickness of about 75 ft. over an even surface, but elsewhere it is found filling hollows in a pre-Cretaceous hilly topography and attaining a local thickness of over 500 ft. The pebbles are composed chiefly of schist, although those of limestone and granite-porphry are not entirely absent. With the continued subsidence of the region, the Morita formation, comprising about 1,800 ft.



of unfossiliferous sandstones and shales, with occasional lenses of sandy limestone, accumulated above the basal conglomerate. Conformably overlying these is the Mural limestone, about 650 ft. in thickness, containing abundant fossils characteristic of the Comanche division of the Cretaceous. Most of these limestones, particularly the lower beds, are thin-bedded and impure, but hard gray massive beds, aggregating about 40 ft. in thickness, occur near the middle of the formation and form a cliff that is a conspicuous topographic feature of the hills north and east of Bisbee. The Mural limestone is conformably overlain by the Cintura formation more than 2,000 ft. in thickness and consisting of alternating sandstones and shales, much like those occurring in the Morita formation. These Cintura beds of the Bisbee group are the youngest stratified rocks exposed in the Bisbee quadrangle. As their upper surface is everywhere one of erosion, their original thickness is unknown. The foregoing Cretaceous strata were first described by Dumble,<sup>4</sup> and by him called the "Bisbee beds."

#### *Post-Cretaceous Deformation and Intrusions.*

The beds of the Bisbee group have been deformed by folding and faulting, and are cut by a few small dikes of diorite-porphry, exemplified by the dike at the Glance mine. The folds are generally open, dips of more than 20° being rather exceptional. The general strike is NW-SE., and the prevailing dip northeast. About 7 miles southeast of Bisbee, however, where Paleozoic beds have been thrust by faulting over the Cretaceous, the latter have been turned up steeply, and are in places nearly vertical. East of Bisbee the faults are normal, but southeast of Mule Pass Gulch faults of the reversed or overthrust type predominate. The most striking of these latter faults is that exposed at Gold Hill, 5 miles southeast of Bisbee. The summit of this hill is composed of Paleozoic limestones which have been thrust from the southwest over the beds of the Cretaceous Bisbee group for a distance of at least 2 miles. A view of Gold Hill from the north, showing the Paleozoic limestones thrust from the southwest over the Cretaceous beds, is given in Fig. 4.

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<sup>4</sup> *Trans.*, vol. xxxi., pp. 703-706.

As Tertiary sediments are absent in the Bisbee region, this period was probably marked by the deformation of the Cretaceous and older rocks and by erosion.

### *The Quaternary Valley-Deposits.*

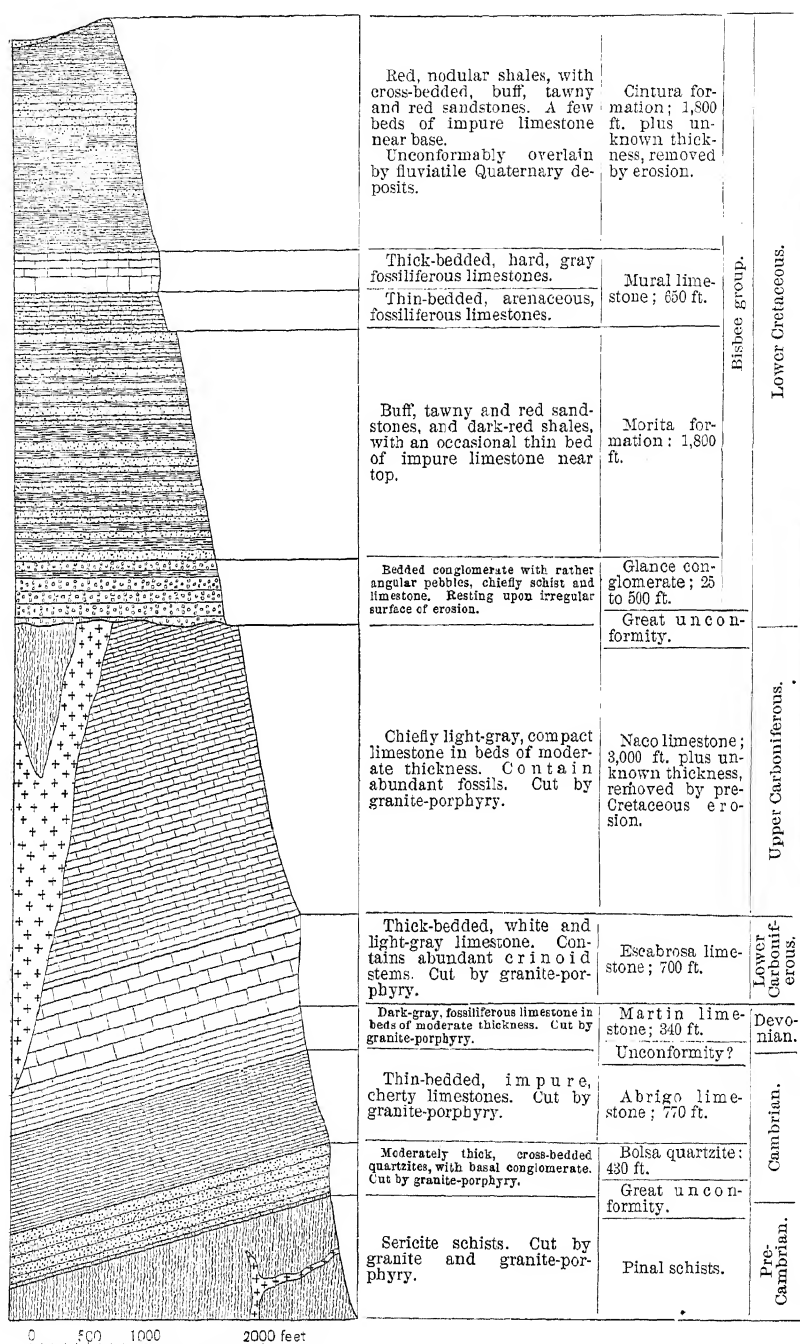
The Quaternary is represented by unconsolidated gravelly deposits flooring the broad valleys that surround the Mule Mountains on the west, south and east. These are in the main fluvial wash, with possibly some finer lacustrine beds at a distance from the mountains.

### *Résumé of Distribution and Structure.*

It is impossible, without the aid of a detailed geological map, to do more than to indicate very crudely the general distribution and structure of the rocks of the Bisbee quadrangle. A NW-SE. diagonal drawn through the quadrangle will pass through the town of Bisbee and form a rough division between the Cretaceous beds on the northeast and the pre-Mesozoic rocks on the southwest. The latter, although folded and faulted, exhibit simple structures and have a prevalent dip to the northeast—away from the older rocks. They undoubtedly once extended over the Paleozoic rocks to the southwest, but have been removed by Tertiary and Quaternary erosion.

In contrast with the Cretaceous beds, the Paleozoic and pre-Cambrian rocks exhibit a highly complex structure, which, if we disregard the undecipherable pre-Cambrian deformation of the crystalline schists, is due to faulting and to intrusions of granite-porphyry, and to folding. In the northwestern part of the quadrangle the Paleozoic beds dip generally to the southwest, but they change near Bisbee to a southeasterly dip, which in turn swings round to a northeasterly dip a few miles southeast of the town. The pre-Cambrian schists, which are extensively exposed in the northern part of the district, pass gradually beneath the Paleozoic beds to the southwest, being less and less frequently exposed in the various fault-blocks and finally disappearing altogether toward Naco Junction. The general stratigraphic relations and sequence of the rocks of the Mule Mountains is shown graphically in the accompanying columnar section, Fig. 6.

FIG. 6.



GENERALIZED COLUMNAR SECTION OF THE ROCKS OF THE BISBEE QUADRANGLE.

## III. ECONOMIC GEOLOGY.

*History of Mining Development.*

Prior to the year 1880, Bisbee was an unimportant lead camp, a single furnace being then in operation upon cerussite mined from the Hendricks claim, close to the town. The copper-ore of the Copper Queen mine was discovered early in this year and was profitably exploited until 1884. This ore was free from sulphur and had an average tenor of 23 per cent of copper. It was treated in two 36-in. furnaces, which in spite of their small size were able with wood as fuel to turn out about 500,000 lb. a month. In 1882, the men composing the present Copper Queen Consolidated Mining Company bought the Atlanta claim near the original discovery, and began prospecting.

In 1884, the Copper Queen ore-body, which had been worked for 300 ft. down an incline, was exhausted. The outlook was gloomy and work was almost abandoned, when a second ore-body was simultaneously discovered from the original Copper Queen incline and from the Atlanta workings. In order to avoid legal complications, the two companies combined as the Copper Queen Consolidated Mining Co., which gradually absorbed the neighboring properties by purchase. In 1886, the old smelting-plant became inadequate and was rebuilt. Greater economy was necessary as the average tenor of the ore in copper had fallen to about 8 per cent and the price of copper had also notably declined.

Shortly after 1890, the completely oxidized ores showed signs of failing. But in 1893 the works were remodeled by the introduction of converters, and sulphide and oxide ores have since that time been successfully worked together by the matte process. The introduction of these converters was due to Dr. James Douglas and marked the beginning of a new epoch in the smelting of copper-ores in Arizona.

Up to the end of 1902, practically all of the copper from Bisbee was the product of the connected group of mines owned by the Copper Queen Consolidated Mining Co. Recently, however, extensive ore-bodies have been opened up in the Calumet & Arizona mine, and in the latter part of December, 1902, this company was turning out from 30 to 40 tons of copper per day from its new smelter at Douglas.

The town of Douglas, situated in the middle of Sulphur Spring Valley on the International boundary, has sprung up with remarkable rapidity during the past year. Its growth is due to the erection here of the new smelters for the Copper Queen, and Calumet & Arizona companies, and to the fact that it is the junction point of the newly completed El Paso and Southwestern Railroad with the Nacosari Railroad, into Mexico. It will undoubtedly become an important smelting point, not only for the Bisbee ores, but for those from Mexico.

*Production*—From August 1880 to the end of 1902, the total output of the Copper Queen Consolidated Mining Co. was 378,047,210 lb. of copper. The product of the recently opened Calumet & Arizona Mine within this period is 2,066,647 lb., so that the total recorded production of the district is 380,113,857 lb. of copper. The maximum output was in 1901, when the Copper Queen mines produced 39,781,333 lb. of copper.

#### *Present Underground Development.*

The Bisbee quadrangle contains two large productive mines, the Copper Queen, and the Calumet & Arizona. In addition to these are a few undeveloped mines that have produced comparatively small quantities of good ore, and a number of prospects ranging from shallow pits to shafts several hundred feet deep connected with extensive systems of drifts.

The Copper Queen mine comprises a maze of workings connecting with the surface through four working-shafts and several less used openings. The area, more or less thoroughly intersected by drifts and cross-cuts, covers nearly half a square mile, where the extreme vertical range of underground exploration is about 950 ft. The workings are practically all in Carboniferous limestone. From 650 to 700 tons of ore are raised per day, chiefly through the Spray, Holbrook and Czar shafts.

The underground development of the Calumet & Arizona mine is confined to the Irish Mag claim. The mine is operated through the Irish Mag shaft, 1,200 ft. in depth, situated 900 ft. south-southeast of the Spray shaft, and about three-quarters of a mile in the same direction from Bisbee. The workings are all in Carboniferous limestone. Stopping in this mine began in the autumn of 1902, and about 300 tons of ore were raised per day before the end of the year.

*General Character of Mineralization.*

The Bisbee quadrangle owes its economic importance exclusively to the occurrence of ores of copper. The unimportant deposit of lead carbonate in Hendricks gulch, which was the first discovery in the district, is still being worked, in a small way, and the Easter Sunday mine has supplied the siliceous gold-ore used in the Copper Queen smelter for converter-lining, but these, so far as known, are the only instances in the quadrangle of ores commercially exploited for other metals than copper.

For a district which has produced nearly 400,000,000 lb. of copper, the Bisbee quadrangle, in spite of the general bareness of its rocky slopes, exhibits little superficial evidence of its mineral wealth. The porphyry mass of Sacramento Hill, southeast of Bisbee, and the schists that partly enclose it, show considerable alteration and contain abundant disseminated pyrite which by oxidation imparts a rusty stain to the rocks, but, so far as known, no deposits of value occur wholly within these discolored rocks. Along the Dividend fault in Bisbee, outcrop some dark rusty masses composed principally of limonite. Similar ferruginous ledges occur in Hendricks Gulch, on Queen Hill and in the limestone south of Bisbee. Experience has shown that such limonitic croppings, although rarely exhibiting superficial indications of the presence of copper, are nevertheless frequently associated with an underlying ore-body. They evidently result from the oxidation of pyrite, the less soluble iron oxide and some silica remaining near the surface, while such copper as was originally present has been carried down by percolating solutions and redeposited at lower levels. These bodies of limonite when not too siliceous are the best surface indications of ore that the district affords. Many of the most important ore-bodies, on the other hand, would have remained undiscovered were superficial phenomena alone relied upon to suggest exploration.

Although the rocks of the quadrangle are seamed with faults and dikes, none of the workable ore-bodies occur as lodes or fissure-veins. With a few exceptions they are irregular replacements of limestone. Originally pyritic, and containing probably subordinate amounts of chalcopyrite, they owe their

present value to secondary concentrations effected by processes of sulphide enrichment and of oxidation.

*Occurrence of the Ores.*

The principal bodies of copper-ore all lie south of the town of Bisbee, within a radius of a mile. They occur in Carboniferous limestone, on the southwest side of a great fault (the Dividend fault), and closely associated with an intrusive mass of granite-porphyry. In the absence of the geological map and sections, the structural relations may perhaps be most clearly presented by a homely illustration. If the half of a broken saucer be placed flat on the table with the fractured edge lying about WNW., and if the back of a book be laid against this edge, we shall have a rough illustration of the geological structure near the town of Bisbee. The saucer represents the synclinal attitude of the Paleozoic beds, from the upper Carboniferous Naco limestone down to and including the Cambrian Bolsa quartzite. The broken edge of the saucer is the great fault, while the book is pre-Cambrian Pinal schist, against which Naco limestone has been dropped by this fault, with a throw of more than 1,500 ft.

The town of Bisbee lies on the fault line. The hills northeast of the town are composed of pre-Cambrian schists. Those just south of it are upper Carboniferous limestone, with lower Carboniferous, Devonian and Cambrian beds coming successively to the surface along the fault to the northwest.

A little less than half a mile southeast of the center of town, the fault encounters a mass of altered granite-porphyry and as a simple fracture disappears. This porphyry, which forms Sacramento Hill (Fig. 5), a local landmark, is a very irregular stock about a mile in diameter. It has invaded the pre-Cambrian schists on the northeast, and the upper Carboniferous (probably also the deeper lying older Paleozoic beds) on the southwest. The available evidence indicates that the intrusion of this porphyry took place after the dislocation of the invaded rocks by the great fault. The latter probably continues to the southeast of the porphyry mass, but it is concealed in this direction by the younger, Cretaceous beds. The Paleozoic beds forming the faulted syncline are not merely flexed but are cut by many faults, some of them of consider-

able throw. These faults are, as far as seen, of the normal type.

The ore occurs very irregularly as large masses within the limestone. The horizontal extent of these bodies is usually much greater than the vertical. They are rudely tabular in form and lie generally parallel to the bedding planes of the limestone. As a rule the important ore-bodies have been found within a distance of 1,000 ft. of the main porphyry mass or of the great fault-fissure just northwest of the porphyry. In the Czar workings of the Copper Queen mine, partly under the town of Bisbee, ore-bodies have been worked from the surface down to a depth of about 400 ft., but toward the southeast the bulk of the ore occurs at increasing depths. In the Calumet & Arizona mine, about 3,500 ft. to the south of the Czar, no large ore-bodies were encountered until the shaft had penetrated about 800 ft. below the level at which the first ore-body was discovered on the Copper Queen claim. The ore thus occurs at increasing depths toward the center of the local synclinal basin. Detailed structure-sections will probably show, however, that the upper limit of the ore increases in depth somewhat less rapidly than would be the case did it correspond to a definite stratigraphic horizon.

With the exception of the extreme western part of the Copper Queen mine, all of the productive and important workings in the vicinity of Bisbee are in the Carboniferous limestones. It is probable that the greater part of the ore-bodies occur in the granular limestones of the lower Carboniferous, but the distinction between upper and lower Carboniferous beds can rarely be satisfactorily made underground. Some important ore-bodies certainly occur in the lower part of the upper Carboniferous. On the other hand, no ore-bodies of consequence have yet been found in the deeper-lying Devonian and Cambrian limestones. In the Copper Queen mine, local usage has distinguished an "upper lime" and a "lower lime." As far as could be seen, however, this distinction is largely imaginary and is based on no constant lithological or structural features. The "lower lime" appears to be any limestone lying underneath the known ore-bodies. It is in the main lower Carboniferous, and the ore-bearing possibility of the underlying and Devonian and Cambrian beds are yet to be ascertained by deeper prospecting.



Although the ore-masses in general are what are usually termed "flat" ore-bodies, dipping gently with the enclosing beds, they are related to other structures as well as to bedding planes. Ore is usually found in large masses along the contact of the limestones with the main porphyry mass. This contact, however, has not been thoroughly explored, and much of the ore along it consists largely of low-grade, partly oxidized pyrite. Dikes and sills of porphyry occur in the limestones at various distances from the main intrusive mass and these are almost invariably associated with ore in the adjacent limestone. In some cases large ore-bodies followed for a long distance in the general plane of the bedding have been known to turn down almost vertically alongside a porphyry dike. Fissures in the limestone have also undoubtedly influenced the distribution of the ore.

While the main porphyry mass of Sacramento Hill is often heavily impregnated with pyrite it has not been shown to contain workable ore-bodies. It is certain, moreover, that many of the porphyry dikes encountered in the workings of the Copper Queen mines show little mineralization even when in contact with ore.

### *Mineralogical Character of the Ores.*

The ores worked by the Copper Queen Consolidated Mining Co. up to 1893 were oxidized ores, consisting chiefly of malachite, azurite and cuprite, and native copper. In the upper levels the malachite and azurite occurred as beautiful incrustations and stalactites lining caves in the limestones. These "caves" have been exhausted, and although oxidized ore is still abundant, it occurs generally as soft earthy masses, often containing cuprite and native copper, and usually associated with large amounts of limonite and kaolin. Native copper and crystalline cuprite are still abundant in the recently opened workings of the Calumet & Arizona mine.

The original sulphide ores, from which the oxidized ores have been derived, consist of pyrite containing variable amounts of chalcopyrite and sometimes a little sphalerite. These pyritic ores are sometimes directly in contact with oxidized ores, but it is not uncommon to find masses of chalcocite between the two. This mineral, familiarly known as "copper

glance," or "glance," is the most important sulphide occurring in the Bisbee quadrangle, since nearly all of the bodies of workable sulphide ore owe their value to its presence. So far as observed, it occurs only in massive form, distinct crystals being nowhere seen. It is sometimes firm and compact in texture, but is often rather soft and occasionally has almost a sooty character.

The chalcocite occurs at various depths, but nowhere far from ore showing at least partial oxidation. Its most characteristic place of occurrence is in the irregular zones of rich sulphide ore that usually intervene between masses of lean pyrite and oxidized ores containing cuprite, native copper and carbonates. The ore of these zones often consists of soft crumbling pyrite, which is intimately associated with dull, cryptocrystalline chalcocite in the form of a thin envelope about the pyrite grains. The greatest depth at which chalcocite has been observed is 1,050 ft., in the Calumet & Arizona mine. Bornite has been reported from some of the ore-bodies, but arsenical or antimonial compounds of copper are absent, so far as known. The copper-bearing minerals of the oxidized ores include malachite, azurite, chrysocolla, melanochalcite, aurichalcite, brochantite, cuprite, tenorite and footeite.

Of the various gangue minerals associated with the ores, calcite, in its rôle of principal constituent of the limestone in which the important ore-bodies occur, is the most abundant. It is seldom, however, that the mineral forms so large a part of the altered limestone in the immediate vicinity of the ore as it does of the unchanged rock in which mineralization has not been active. It has largely been replaced in the process of mineralization by pyrite, amphibole, pyroxene, garnet, chlorite, quartz, vesuvianite and other minerals. The resulting rock is usually of more compact texture than the unaltered limestone and is slightly greenish in tint. The metamorphic silicates are, as a rule, recognizable only under the microscope, and the contact-metamorphism is therefore very inconspicuous as compared with the altered limestones of the Cananea district to the south.

### *Genesis of the Ore-Deposits.*

Two general processes have operated to form the ores now exploited. These are (1) metasomatic alteration, including py-

ritic mineralization, and (2) oxidation and its attendant phenomena of transportation and enrichment. Concerning the precise boundary between these two general activities some difference of opinion is possible; but as regards the essential share of each in the genesis of the ores there can be no question. There are few known ore-bodies in the Bisbee quadrangle which do not demonstrably owe their value to the co-operation of both processes. This was also the conclusion reached by Emmons<sup>5</sup> from the descriptions of Douglas and from a brief visit to the Copper Queen mine. The existence of workable masses of ore that have resulted wholly from the primary metasomatic mineralization, while not denied, is still as far from proof as when Dr. Douglas published his excellent account of the Copper Queen mine about three years ago.

As shown in those underground workings which are below the intermediate zone penetrated by oxidizing reagents, the sulphide minerals ascribable to the early period of metasomatic alteration are common pyrite, and perhaps also a little chalcopyrite and sphalerite. The occurrence of some cerussite in the limestones suggests that galena may also occur, although this mineral has not been seen. Associated with these sulphides are amphibole (tremolite), pyroxene (diopside), garnet (grossularite), vesuvianite, quartz and chlorite, in the limestones; quartz, sericite, chlorite, perhaps kaolin, and a little epidote in the granite-porphry; and quartz and sericite in the Pinal schist.

The metamorphic minerals are not distributed uniformly through the limestone, but are arranged in zones about the porphyry stock of Sacramento Hill. Nearest the porphyry the former limestone is now a fine-grained aggregate consisting chiefly of quartz and calcite. The width of this siliceous zone may be roughly estimated at about 200 ft. It is probably very irregular and is not sharply defined at its outer border. Although heavily impregnated with pyrite, it has not proved such favorable ground for ore as the next zone to be described.

Encircling the siliceous zone are the pale-green altered limestones in which pyrite, tremolite, diopside, grossularite and probably vesuvianite are the characteristic minerals. In this zone quartz is comparatively rare. Calcite, however, is abun-

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<sup>5</sup> "The Secondary Enrichment of Ore-Deposits," *Trans.*, vol. xxx., p. 192.

dant, and may make up half or more of the altered limestone. At its inner edge, this zone grades somewhat indefinitely into the siliceous zone. About its circumference it passes by obscure gradations into the unmetamorphosed limestones of the region. The rock of this zone of metamorphic silicates is softer than that of the inner siliceous zone. It is frequently traversed by little irregular surfaces of shearing, so that the rock tends to break along many curved surfaces into small fragments. The width of this zone cannot be stated with precision, on account of the indefinite character of its boundaries, but it may be very roughly estimated at about 1,000 ft. The zone is not, however, limited in plan to a simple concentric band about the porphyry, but extends for a considerable distance to the northwest, alongside the Dividend fault.

In general, the zone in which the metamorphic silicates are developed is that in which most of the ore-bodies are found. In the western part of the Copper Queen mine, however, extensive ore-bodies occur in limestone that is perhaps outside of this zone. In other words, the deposition of metallic sulphides appears to have had a wider range, and to have extended farther from the porphyry into the fissured limestone than did the formation of the metamorphic silicates.

That the original ore-deposition was genetically connected with the intrusion of the granite-porphyry of Sacramento Hill is reasonably certain. The other factors, which by their fortunate conjunction determined the original mineralization of this part of the district, were the contact of this considerable mass of porphyry with limestones, the occurrence of the Dividend fault and other less important fractures, and the attitude of the limestone beds with reference to the porphyry and to these faults. The hypothesis is advanced that the deposition of the cupriferous pyrite and the metamorphism of the limestone, schist and porphyry were effected by the mingling of solutions from different sources in the vicinity of the Dividend fault and of the Sacramento Hill porphyry. The principal function of the porphyry is believed to have consisted in the supply of heat to such solutions as rose from great depth through the fault fissures and in determining the locus of chemical activity that resulted in the deposition of the ore.

The source of the ore-materials is not known. They may have risen through the Dividend fault from depths far below

the bottom of the syncline of Paleozoic rocks. They may have been collected by solutions moving through one or more of the Paleozoic limestones on their way down to the locus of deposition. Lastly, they may have been derived from both sources.

Few, if any, of the ore-bodies now known and worked in the Bisbee district were formed directly by the original mineralization just discussed. That process resulted, so far as underground workings show, mainly in the formation of bodies of cupriferous pyrite passing peripherally into metamorphosed and pyritized limestone. These bodies contained originally too little copper to be classed as ore, as that term is defined by present economic conditions of working. Further concentration was necessary, and the profitable ores of to-day have been derived from these pyritic masses by secondary processes involving general oxidation, with all of its attendant chemical phenomena, transfer of material, and the local enrichment of certain portions of the sulphide masses by the formation of chalcocite, and, in some cases, of chalcopyrite.

The lower limit of oxidation of the Bisbee ores is very irregular and is apparently uncontrolled by any constant groundwater level. In the Calumet & Arizona mine oxidized ores occur at a depth of 1,000 ft., while residual masses of sulphide ore occur in the adjoining Copper Queen mines within 150 ft. of the surface. Masses of lean pyrite are sometimes completely enclosed in an envelope of high-grade chalcocite and oxidized ores.

#### IV. FUTURE OF THE DISTRICT.

Although more or less mineralization occurs at many points in the Mule Mountains, there is little to indicate that any deposits of copper-ore will ever be found in the Bisbee quadrangle approaching in importance those already known and probably awaiting discovery in the faulted limestone syncline about Sacramento Hill. For more than 20 years the Copper Queen mine has produced an average exceeding 16,000,000 lb. of copper annually. Recently, the Calumet & Arizona Co. has begun energetic operations in ground almost surrounded by the property of the Copper Queen Consolidated Mining Co. Not only is there sufficient ore known in these mines to keep them in operation for many years to come, but there is no clear evidence that the bottom of the ore-bearing ground has been reached in any of these extensive workings. Moreover, the

statement may be ventured that the specter of the "lower lime" has hitherto had an undue influence in restricting prospecting to nearly horizontal planes. There is certainly a reasonable hope of finding ore-bodies in the Martin and Abrigo limestones beneath the masses that have been so profitably worked in the overlying Carboniferous beds. The occurrence of small bunches of ore in the Abrigo limestone at the Whitetail and Wade Hampton claims shows that ore-deposition may take place in these lower beds. It may be, that owing to their greater depth, such ore-bodies if discovered will be found to consist of low-grade cupriferous pyrite, unenriched by the generally descending solutions that have contributed so largely to the value of the known masses. This, however, is a point that prospecting alone can determine. It would seem that near the Dividend fault the conditions are favorable for the continuance of enriching processes to depths greater than those now reached in this part of the underground workings.

But more than this, it may be pointed out that less than half of the semicircular mineralized zone about the porphyry mass of Sacramento Hill has been explored at all. Ore was first discovered at the surface on Queen Hill at the northwest end of the zone. From this discovery, developments have been pushed by underground exploration to the south, often with little or no surface indication of ore. There still remains, however, an extensive area of unknown but promising ground, lying just south of Sacramento Hill and extending eastward toward the southeastern continuation of the Dividend fault, which is here concealed by the Glance conglomerate. This is the eastern half of the semicircular mineralized girdle about the intrusive mass of porphyry. While structurally the beds south of Sacramento Hill form a less favorable nook for the deposition and concentration of ores than that enclosed between the Czar and Dividend faults and the northern contact of the porphyry, yet their attitude with reference to the intrusive stock and to the probable continuation of the Dividend fault down Mule Gulch is distinctly favorable to mineralization. The developments in the Spray and the Calumet & Arizona mines have shown that profitable ore-bodies are by no means confined to the structural pocket in the vicinity of the Czar and Holbrook shafts. There is no known reason why they should not yet be found skirting the southern contact of the porphyry for some distance east-

ward towards Mule Gulch. Recent exploration with diamond drills shows that a considerable mass of porphyry extends southward from the Sacramento Hill stock beneath the Glance conglomerate of Mule Gulch. It is thus probable that the ore zone, if present, also turns south along the contact of the limestones with the porphyry, and does not extend eastward under Mule Gulch itself.

It is true that the easterly dip of the beds south of the porphyry stock carries the Escabrosa limestone to a much greater depth near Mule Gulch than it has in any of the present workings. It is probable that this depth, a short distance east of the Gardner shaft, may be so great as to preclude the occurrence within the Escabrosa limestone of any but original, unenriched pyritic ore, which may or may not be workable. But judging from surface indications, there has been considerable mineralization of the Naco limestone south of Sacramento Hill, and there is nothing improbable in the occurrence of high-grade ores in this limestone at stratigraphically higher horizons than those in which ore-bodies have hitherto been found. The exploration of this ground calls for no greater outlay or boldness than has already been displayed in other parts of the district with less assured hope of reward.

The outlook for finding profitable ore-bodies within the main porphyry stock of Sacramento Hill is not regarded as particularly promising. As the dump of the Copper King shaft shows, this rock may be very heavily impregnated with pyrite. But the mineralization seems to tend more to abundant dissemination than to the formation of solid sulphide masses. Even if the latter occur, it is doubtful whether they would be found sufficiently cupriferous to constitute ore. Moreover, the chemical and physical character of the porphyry render it very much less favorable than the limestones to the deep oxidation and secondary concentration that has played so important a part in the genesis of the known ore-bodies. Very little exploration of the porphyry has been made, however, and work in this direction cannot be condemned as altogether vain.

In conclusion, it may be said that Bisbee is less likely to suffer from a lack of ore than from a too rapid exhaustion of those high-grade oxidized ores which are necessary for the economical smelting, by present processes, of the low-grade sulphides.

## The Yellow-Ocher Deposits of the Cartersville District, Bartow County, Georgia.\*

BY THOMAS LEONARD WATSON, GRANVILLE, OHIO.

(New York Meeting, October, 1903.)

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### INTRODUCTION.

KNOWLEDGE of the existence of yellow-ocher deposits in Bartow county, Georgia, dates back to the early mining of manganese- and iron-ores in the vicinity of Cartersville. Systematic mining of the ocher in this area really began, however, probably not longer than a decade back, although the date of its first working is as early as the year 1877. The district has been, for a number of years, one of the principal producers of yellow ocher in the United States; and at no period of its development have mining activities been greater and more successful than at present.

Systematic field-study of the ocher- and manganese-deposits of Georgia was begun by me while I was a member of the

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State Geological Survey, late in the season of 1900; and completed during the field-season of 1902. Separate reports on these deposits are in course of preparation, and as soon as completed they will be published as bulletins of the State Survey. It is the object of this paper to present some of the more important facts brought out in the field and laboratory study of the ocher-deposits of Bartow county, Georgia.

#### HISTORICAL STATEMENT.

The first authentic record of ocher mined in Bartow county, Georgia, was in the year 1877, when Mr. E. H. Woodward began mining it on a property located near the limits of the town of Cartersville. The crude ocher as mined was hauled in wagons to Cartersville and there prepared for market. Mr. Woodward was engaged at the same time also in mining manganese-ores on the Dobbins property, six miles northeast of Cartersville.

Mining of the ocher on a small scale was continued on this and the adjoining property until 1890, when the Georgia Peruvian Ocher Co., supported by Western capitalists, became the owner of the property, and improved methods in the preparation of the ocher for market were introduced. Hauling to Cartersville was discontinued on account of the roads, and a plant for preparing the ocher was located at Emerson, two miles south of the mines. The first shipment of American ocher to Europe is reported to have been made in December, 1890, from the Cartersville mines, a consignment of 50 tons having been shipped to England.<sup>1</sup>

In 1890, two experienced ocher-men from the East, Mr. J. C. Oram of Vermont and Mr. E. P. Earle of New York, became interested in the company, and modern machinery and improved methods were brought into use. Mr. Oram was the first to introduce in the district the natural process of air-drying (sun) in vats dug in the ground. The ocher industry in Georgia properly dates from this year.

At present, four plants of large size and thoroughly equipped are engaged in mining and shipping the ocher in the Cartersville district: namely, the Georgia Peruvian Ocher Company;

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<sup>1</sup> *Mineral Resources of the United States*, 1889 and 1890, p. 509.

the Cherokee Ocher & Barytes Company; the Blue Ridge Ocher Company; and the American Ocher Company. The last plant was added in 1902. In every instance the mill for preparing the ocher is located at the mines, thereby reducing the cost of production from what it was in the early period, when mine and mill were separated by a considerable distance. The present plants are all located within two-and-a-half miles of the town of Cartersville, which is the shipping point.

Several unsuccessful attempts have been made to produce ocher at Rockmart, in the adjoining county, Polk, on the southwest, but each time the venture was abandoned.

#### GEOLOGY OF THE DISTRICT.

The area here described, as shown on the accompanying map, Fig. 1, is limited to the southeastern portion of Bartow county, in northwestern Georgia; and it lies in the vicinity of the town of Cartersville, from which the district derives its name. It is one of the most productive ore-districts in the Southern Appalachians. Stratigraphically, the area is nearly equally divided between the Paleozoic formations on the west and the older crystalline metamorphic rocks of the Piedmont plateau and the Appalachian mountains on the east. The irregular line separating the two groups of unlike rocks marks the position of the Cartersville fault, which is the most important structural feature in the region.

As indicated in Fig. 1 (C, C, C, C), the rocks belong to two geologically distinct groups, which show marked differences as to age and kind. To the west of the fault-line the rocks are sedimentaries, and include quartzites, sandstones, shales, and limestones of Cambro-Silurian age. To the east of the fault-line the rocks are metamorphic crystallines, derived in part from original igneous masses and in part from original sediments. Over parts of the area it is difficult to state definitely whether the original rock was igneous or sedimentary in origin. The altered sediments include conglomerates, slates, schists, and probably some of the gneisses. Wide variation in composition, and to a less extent structure, characterize the intrusive rocks, ranging from extremely basic rocks of the diabase type to acid granites, with perhaps diorite as the most common type. These igneous masses are no longer composed



of rocks massive in structure, but, on account of intense pressure-metamorphism, they are now rendered highly schistose. On account of the absence of fossils in the sedimentaries of this series, and because of the rocks having every appearance of extreme age, Hayes has grouped them as Algonkian (Ocoee) in age.

The rock-sequence for the Paleozoic formations on the west side of the fault-line, named in descending order, becomes :

Silurian, . . .	4 Knox dolomite.
Cambrian, . .	$\left\{ \begin{array}{l} 3. \text{Rome and Conasauga shale.} \\ 2. \text{Beaver limestone.} \\ 1. \text{Weisner quartzite.} \end{array} \right.$

Of these formations only the Weisner quartzite is ocher-bearing and of importance in the present connection, hence no description of the other formations will be given in this paper.

### *The Weisner Quartzite.*

The yellow-ocher deposits of the Cartersville district are limited exclusively to the Weisner quartzite. As shown on the map, the principal area of the quartzite forms a continuous narrow belt, approximately 15 miles long and several miles wide, in the central portion of the area mapped. Its eastern limit is the Cartersville fault, which marks the contact of the formation on the east with the rocks of the Ocoee series. Near the middle western margin of the map, faulting has exposed two additional narrow strips or bands of the quartzite, which are marked by the entire absence of ocher. As nearly as can be estimated the thickness of the quartzite in this area does not exceed 2,000 feet.

Lithologically, the formation is not entirely uniform, but in places it shows considerable variation, both in composition and texture, and in color as well. It is composed principally of a compact, fine-grained vitreous quartzite varying from light to dark gray in color, and in places containing beds of fine-grained conglomerate. Intercalated beds of drab to darker colored siliceous shales of varying thickness, much crumpled, contorted and altered in places, are often met with. The formation contains much pyrite in the form of grains and crystals in places,

and this mineral seems to be equally abundant in both the quartzite proper and the interbedded shale layers. The two mineralogically-unlike beds, shale and quartzite, are likewise ocher-bearing, and the difference in composition of the rock serves as a basis for making two grades of the ocher. The ocher found replacing the shales is prevailingly darker in color, because of the large proportion of admixed argillaceous or clayey matter derived from the shales, which cannot be separated from the refined ocher; while the ocher found replacing the quartzite proper is uniformly lighter in color, because of less admixed clay.

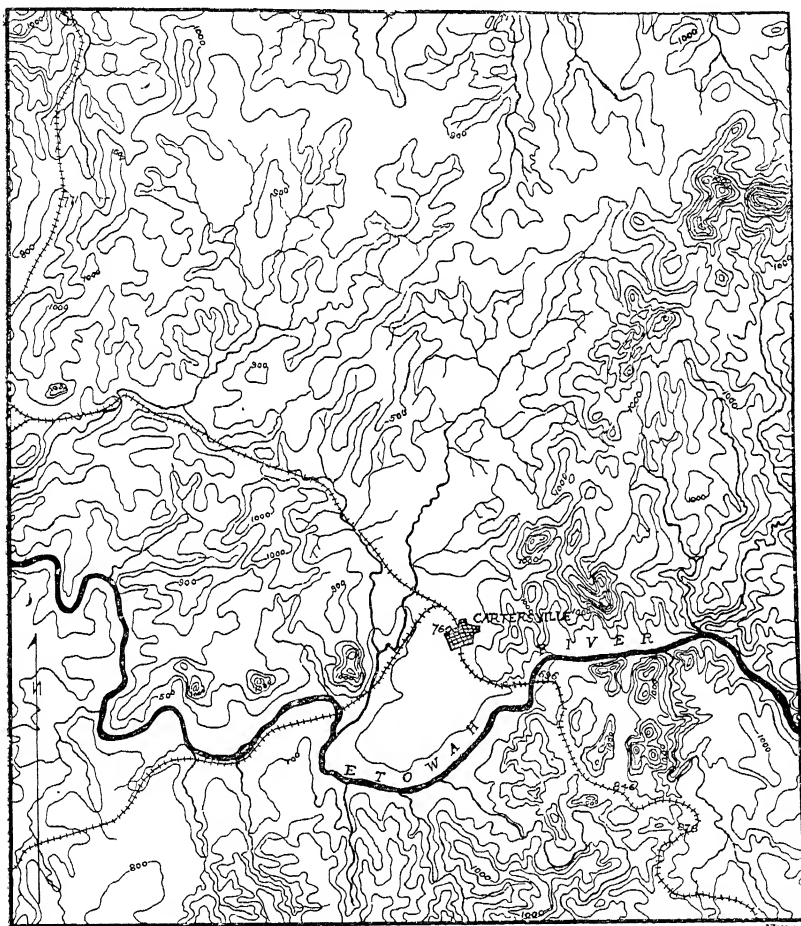
The effects of intense pressure-metamorphism are plainly evident in all parts of the quartzite formation, as shown in Fig. 3. As a result of the action of the compressive forces the quartzite layers have been sharply folded, and in addition the formation has been extensively crushed and shattered over most of its parts, especially in the ocher-bearing portions. So extensively crushed and shattered is the quartzite as indicated in some of the sections of the larger ocher-openings in the area, that it is almost impossible to determine the original bedding of the rocks. As often shown by its brecciated condition the quartzite is presumably cut by numerous faults, but such lines of fracture, if they exist, have not been traced. As Hayes has stated, the physical forces were probably attended by increased chemical action, inferred from the formation of the ocher.

The composition of the quartzite proper is shown in the chemical analysis below made by the N. P. Pratt laboratory, in Atlanta, Georgia, of specimens of the rock collected by me from different exposures of the formation.

	Per Cent.
Silica, . . . . .	90.36
Alumina, . . . . .	1.52
Iron sesquioxide, . . . . .	0.57
Iron disulphide, . . . . .	1.50
Lime, . . . . .	0.27
Magnesia, . . . . .	0.27
Soda, . . . . .	0.43
Potash, . . . . .	0.16
Manganese oxide, . . . . .	none
Titanic oxide, . . . . .	0.07
Barium sulphate, . . . . .	4.46
Water, . . . . .	0.31
Total, . . . . .	99.92

Attention is called in the analysis to the percentages of iron disulphide (pyrite) and barium sulphate (barite) present in the rock. The occurrence of pyrite in the rock has been referred to above. The mineral barite is present to a greater or less

FIG. 2.



0 1 2 3 4 5  
Scale of Miles.

TOPOGRAPHICAL MAP OF THE CARTERSVILLE DISTRICT, GEORGIA. (Cartersville Topographic Sheet, U. S. Geological Survey.) Contour-interval, 100 ft.

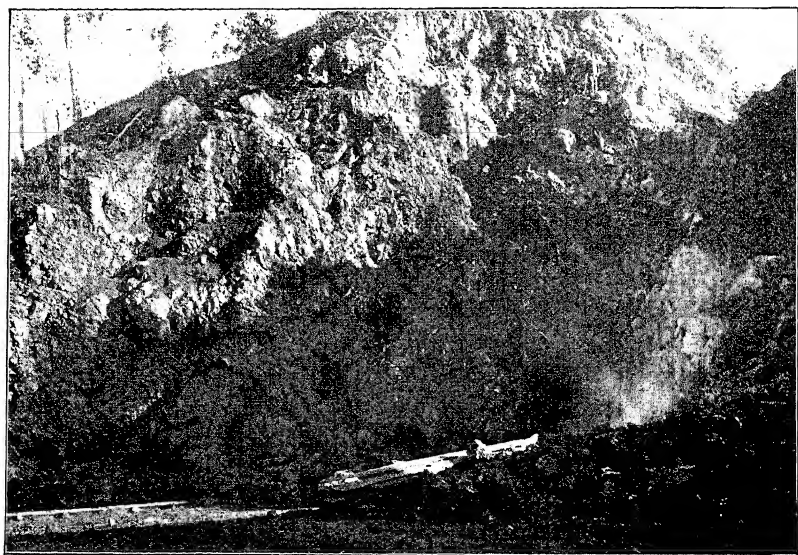
extent in all the ocher-deposits of the district, but its presence in the fresh rock as a mineral constituent is nowhere indicated, either macro- or microscopically. The very small percentages of lime and alkalis shown in the analysis confirms the micro-

FIG. 3.



CUT ALONG WESTERN & ATLANTIC R.R., 1.5 MILES SE. OF CARTERSVILLE, SHOWING FOLDED AND CRUSHED WEISNER QUARTZITE.

FIG. 4.



PART OF SECTION THROUGH QUARTZITE RIDGE, EXPOSED BY THE ETOWAH RIVER, AT THE WOODEN BRIDGE, 2 MILES FROM CARTERSVILLE.

FIG. 5.



OCHEP-PIT BETWEEN TILTED AND CRUSHED QUARTZITE ALONG ROWLAND SPRINGS ROAD ON THE BLUE RIDGE CO.'S PROPERTY.

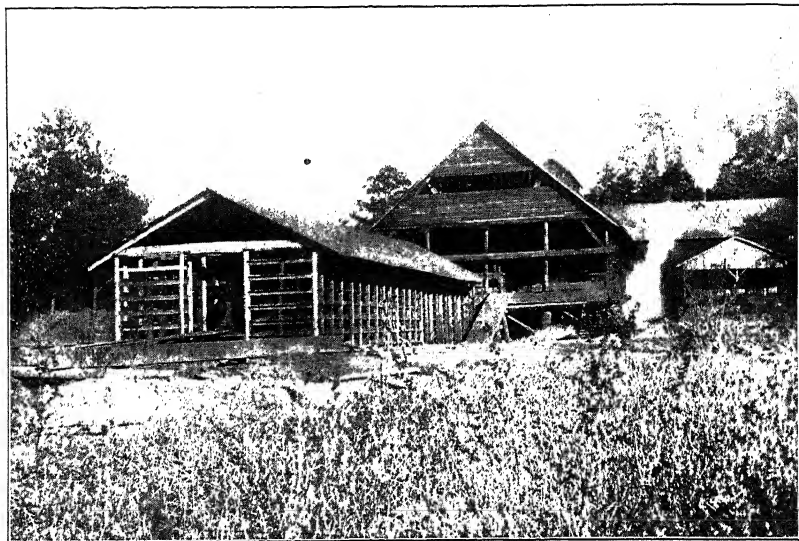
FIG. 6.



OCHEP-PITS IN QUARTZITE, 1.5 MILES SE. OF CARTERSVILLE. THE QUARTZITE-LAYERS ARE NEARLY COVERED WITH CRUMBLING ROCK.

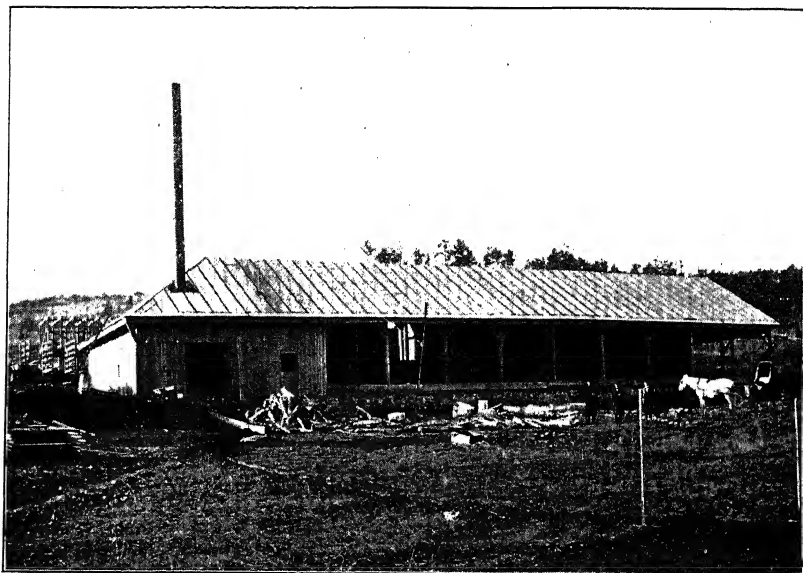


FIG. 7.



PORTION OF AN OCHEE-PLANT, NEAR CARTERSVILLE, SHOWING DRYING-SHED AND RACKS.

FIG. 8.



PLANT OF THE AMERICAN OCHEE COMPANY, 2.5 MILES N.E. OF CARTERSVILLE.

scopic study of a large number of thin sections of the rock, in the nearly complete absence of feldspars. A few thin sections, however, showed sporadic grains of both microcline and a striated plagioclase.

### *Topography.*

With respect to surface configuration, the district may be divided into two nearly equal, unlike areas, as indicated on the accompanying map, Fig. 2. The line separating these two areas is an irregular one roughly paralleling the fault-line, and located from one to three miles west of it, and marking the contact between the Weisner quartzite and the Beaver limestone. The contour lines on the map, Fig. 2, mark off quite strongly the dividing-line between the two unlike areas.

That part of the district north of the Etowah river and west of a line drawn northeastward through Cartersville marks a rather smooth plain, etched out of the soft shales and limestone of Cambrian age. Its average elevation above mean sea-level is between 800 and 900 ft. Slight inequalities in the form of irregular hills or minor ridges, rising from 100 to 125 ft. above the general surface of the plain, denote unreduced areas. Elevations of less than 800 ft.—from 750 to 775 ft.—are recorded in places along some of the larger stream-courses. Westward, the plain grades into the Knox dolomite plateau, a slightly more resistant magnesian limestone, whose general average elevation is but little above that of the Cartersville portion of the Paleozoic plain.

Beginning with, and including, the long central band of Weisner quartzite, that part of the mapped area to the south of the Etowah river and east of the plain already defined, is a second area whose surface is higher than that of the Paleozoic plain described and in marked contrast to it. The larger portion of this area forms the northwestward extension of the Piedmont plateau. Its general surface-elevation will average less than 1,000 ft. above mean sea-level, with numerous residuals in the form of irregular hills and ridges standing several hundred feet above the plateau surface. The surface then is an irregular one, trenched by comparatively deep and narrow stream-channels, in many places cut through the thick covering of decayed rock into the hard rock beneath. The northeast corner of the mapped area forms the equivalent lowering por-

tion of the Appalachians, showing elevations of from 1,800 to 2,000 ft. This marks the roughest surface in the district. The higher and more roughened surface of the eastern half of the mapped area is etched out of geologically old, highly-tilted and disturbed metamorphic crystalline rocks, whose age for the most part is pre-Cambrian.

The entire district covered by the map is well watered. Its drainage is through numerous nearly north- and south-flowing streams tributary to the Etowah river, the master stream of the region, which has a general westward course, passing within a short distance south of the town of Cartersville.

### *Rock-Weathering.*

The rocks of the area are very generally buried under a considerable thickness of residual clays derived from the decay of the underlying rocks by the usual action of the atmospheric agents. Over many parts of the region exposures of the fresh rock are seldom seen. A thickness of 100 ft. and more of the residual decay is frequent over the district. Difference in composition of the sub-terrane is shown by an equally marked difference in the chemical and physical properties of the derived decay. The decay derived from the limestone is usually a deep-red ferruginous clay with or without admixed chert fragments, as the original rock was chert-bearing or not. That derived from the harder and more resistant quartzite is a light gray siliceous clay, in which the proportion of clay is usually relatively smaller than in that derived from the limestone. So strongly marked are the properties of the residual decay derived from the lithologically-unlike sub-terrane, that the areas of originally fresh rock can usually be differentiated and traced with considerable accuracy by its decay.

The decay over the quartzite area is thickest in the valley-bottoms, and thinnest near and on top of the higher and steeper ridges. Along the steeper ridge-slopes exposures of the comparatively fresh and hard rock are not uncommon. On the ridge-tops large reefs and broken masses of the hard quartzite are frequent. The residual decay derived from the quartzite is largely admixed with fragments of various sizes of the quartz-rock in all stages of decay, from partially discolored hard rock to masses of loose or incoherent quartz-grains or sand.

## THE OCHEE-DEPOSITS.

As indicated on the accompanying map, Fig. 1, the ocher-belt has an approximate length of about eight miles in a nearly north-south direction. Traced by the outcroppings and the prospect-openings the belt is a very narrow one, not exceeding perhaps two miles in the widest point. It has its southernmost extension at and to the west of Emerson, about two miles south of the Etowah river, and is traced in a northward direction, about one mile east of Cartersville, to a point north and to the west of Rowland Springs. Beyond this point surface-indications disappear and no prospect-openings have been made, hence it is not possible to state definitely that this marks the extreme limit of the belt to the north.

*Petrography of the Quartzite.*

Some portions of the Weisner quartzite, as previously stated, contain interbedded siliceous shales of dark color. The quartzite proper varies from dense, nearly white, and vitreous massive beds, without distinct evidence of its fragmental character visible to the unaided eye, to massive beds of distinct granular quartzite of light and dark gray colors. By far the majority of the beds are composed of the granular quartzite in which the fragmental character is plainly evident. In the granular type the rock varies from an even-grained, fine-granular quartzite to a distinct conglomerate facies, in which the quartz-pebbles are usually of very small size.

Thin sections show it to be a rather pure quartz rock, composed of quartz-grains of somewhat variable size. In general the larger grains contain some inclusions of foreign mineral matter. Hardly a section examined failed to show, in a number of the larger grains, abundant hair-like needles probably of rutile, which are often bent and curved and in many cases broken. Usually the grains are considerably clouded by very fine innumerable dust-like particles, without definite arrangement, whose exact nature it was not possible to determine. Inclusions of slender prismatic crystals of apatite are not uncommon in some of the thin sections. The general shape of the larger grains is round. When examined in detail the outer margin of the grains invariably presents an irregular, angular outline, formed by the interlocking or dove-tailing of individ-

uals in a surrounding mosaic of much smaller quartz-granules, which fills the entire interstices between the larger grains. This mosaic of finer quartz-particles bounding the larger grains is clearly the result of peripheral shattering from compression, a circumstance which is further confirmed by the general undulous extinction of the larger grains, and by the greatly crushed condition of the formation in the field.

Besides quartz, there occurs in some of the sections a slight sprinkling of feldspar grains, including microcline and a striated plagioclase, some calcite and occasional grains of a titaniferous iron oxide. Next to quartz, pyrite is the most abundant mineral. It frequently occurs as fresh grains and crystals; more often as partially and entirely oxidized in the form of iron oxide. In the latter case the original pyrite may be altered to a limonite pseudomorph, but generally the change has been rapid and a cavity preserving the outline of the pyrite, only slightly stained or partially filled with the iron oxide, forms the only evidence of the former presence of the pyrite. The mineral, both in the fresh and in the completely oxidized or altered condition, is often present in the same thin section. The stain of yellow iron oxide derived from the oxidation of the pyrite discolours the section for some distance around and away from the position of the original sulphide mineral. The staining extends the farthest along the sutures between the quartz-grains and the fracture-lines, which are present at times in the grains.

Microscopic study shows quite plainly the relations of the ocher to the quartzite and its mode of occurrence in the rock. After careful study of a number of thin sections under the microscope of the ocher-stained rock from different parts of the belt, my results accord so closely with Hayes's description that I quote him in full. He says:<sup>2</sup>

"When the transition-rock is examined under a microscope, the character of the transition can be seen even more clearly. The more compact portions, which are only slightly stained with iron, are seen to be composed of a transparent ground-mass, threaded with minute cavities, which penetrate the rock in all directions and contain a fine dendritic growth of iron oxide. The latter occurs only rarely in isolated grains, but generally in clusters of minute grains or fibers, attached to each other and branching irregularly from a central stem. They have

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<sup>2</sup> *Trans.*, xxx., 416 (1890).

no trace of crystal form. Passing toward the ore-body, these minute passages become larger and increase in frequency, until only a finely branching siliceous skeleton remains, the greater part of the rock having been replaced by the iron oxide. Under polarized light, the transparent ground-mass is broken up into an aggregate of small quartz-grains, penetrated in all directions by the iron oxide. The latter does not lie between the individual grains, but passes through them, as though the ground-mass were quite homogeneous. The process of replacement is never complete; for all the ocher contains more or less sand. When this is washed clean from the iron oxide, it is found to differ from ordinary sand-grains in having extremely irregular outlines. This sand, as might be anticipated from the microscopic structure of the slightly altered quartzite, is evidently composed, not of the original grains of the rock, but of detached portions of the irregular siliceous skeleton, which, in the intermediate stages of replacement, holds the iron oxide in its cavities."

A chemical analysis made at the N. P. Pratt laboratory in Atlanta, Georgia, of specimens of the quartzite collected by me is given on page 648.

The ocher occurs only in the fresh Weisner quartzite and its decay; and exposures of the fresh rock indicate that it occupies an extensively shattered zone in the formation. It is found in place in the fresh quartzite and in a similar position in the residual clays derived from the decay of the quartzite. Examples of both occurrences are abundant in the region.

#### *Associated Ore-Deposits.*

Besides ocher, the district is one of the largest and principal producers of the ores of iron and manganese in the State. In addition to these, some barite has been mined and shipped from the area. The latter mineral, while rather largely distributed over portions of the region, has not proved profitable, for the reason that it is rarely sufficiently concentrated for mining alone, and it is not of sufficient purity to make a desirable grade of marketable baryta.

The ores are often very closely associated with each other, but recent study of them shows the genesis of the types of ore-deposits to have been quite different. The deposits of yellow ocher are rarely entirely free from some admixture of one or all of the other types of ores mentioned. These are usually present only in small quantity in the ocher, and generally in sufficiently large fragments to admit of nearly complete separation from the ocher by the usual process of cleansing.

In many cases manganese oxide, in the form of very finely disseminated grains or powder, is not entirely freed from the

refined product, and it is claimed that a faint greenish cast is thereby imparted to the ocher. The deposits of ocher and manganese are frequently in juxtaposition, and the openings employed in working the manganese are now used for mining the ocher. Such occurrence is well illustrated on the property of the Blue Ridge Ocher Company, where large quantities of manganese-ores were formerly mined, and the tunnels and shafts are now utilized for removing the ocher. On the same property a large quantity of hard and porous spongy masses of limonite, occurring as lenses and irregular seams in the ocher beds, was being removed with the ocher during the summer of 1902.

At the Etowah river, on the Georgia Peruvian Ocher Company's property, several miles southeast of Cartersville, large clusters and groups of barite crystals are formed in the pockets of ocher and in the fractures and caverns of the quartzite. The barite often occurs in divergent groups of massive tabular crystals, giving a crested appearance, grading into both straight and curved laminated masses.

*Chemical Composition of the Ocher.*

The chemical composition of both the crude and the refined ocher is shown in the table of analyses below, made by the N. P. Pratt laboratory, in Atlanta, Georgia.

*Table of Chemical Analyses of the Crude and Refined Ocher.*

Components.	Samples.							
	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Fe <sub>2</sub> O <sub>3</sub> .....	72.29	56.29	65.49	54.60	67.37	61.40	67.32	62.79
Al <sub>2</sub> O <sub>3</sub> .....	5.55	10.15	7.20	6.68	6.85	7.14	5.86	6.94
FeO .....	0.46	0.39	.....	.....	.....	.....	.....	.....
MnO <sub>2</sub> .....	0.87	0.54	1.80	1.50	2.04	2.00	.....	.....
SiO <sub>2</sub> (free sand) .....	6.65	8.94	7.76	17.42	6.54	11.89	9.14	6.20
SiO <sub>2</sub> (combined as silicates) .....	3.98	9.49	6.85	10.08	6.61	5.84	6.35	9.78
H <sub>2</sub> O at 105° C. ....	0.55	2.08	0.40	0.48	0.96	0.46	0.78	0.50
H <sub>2</sub> O above 105° C. ....	9.22	11.34	10.50	9.24	9.63	9.37	9.60	.....
Total .....	99.57	99.22	100.00	100.00	100.00	98.10	99.05	.....

I. Crude ocher from Mansfield Brothers property. Lot No. 462, 4th District, 3d Section, Bartow county, Georgia.

- II. Crude ocher from the John P. Stegall property, near Emerson, Bartow county, Georgia.
- III., IV., V. and VI. Refined ocher from the Blue Ridge Ocher Company's property: Lot No. 490, 4th District, 3d Section, Bartow county, Georgia. Furnished by courtesy of the Manager, Captain John Postell, Cartersville, Georgia.
- VII. Refined ocher from the Cherokee Ocher & Barytes Company's property, one mile east of Cartersville. Furnished by courtesy of the President, Mr. T. W. Baxter, Atlanta, Georgia.
- VIII. Refined ocher from the American Ocher Company's property. Lot No. 475, 4th District, 3d Section, Bartow county, Georgia. Furnished by courtesy of the Manager, Mr. Waite, Cartersville, Georgia.

The analyses are sufficiently explanatory, and attention need only be called to one feature, namely, the high percentage of ferric oxide. Assuming all the ferric oxide to be combined with water<sup>3</sup> in the proportion to form limonite, and calculating on this basis the percentage amount of limonite in each analysis, we find an average of 78.33 per cent for the eight analyses. This means that about one-fourth of the entire product consists of foreign mineral matter, in the form of impurities that cannot be separated from the hydrous iron oxide (ocher) by the present methods of cleansing. Field- and laboratory-studies show this admixed mineral matter to consist largely of clay and very finely divided quartz-particles, with, in many cases, smaller amounts of manganese oxide. Notwithstanding these facts the yellow ocher of the Cartersville district is one of the very best mined in the United States, and is the equal of most of the foreign ochers of this color.

#### *Mode of Occurrence of the Ocher.*

As previously defined, the yellow-ocher belt in Bartow county is entirely limited to the Weisner quartzite of lower Cambrian age. Field-study shows that the ocher occurs in both the hard and fresh quartzite and in the residual clays derived from the decay of the quartzite. So far as mining developments have been made in the area, the ocher has nearly equal occurrence in the fresh and in the decayed quartzite. At every point examined, its position in the residual clays is in all respects similar to that in the hard and fresh rock. Its occur-

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<sup>3</sup> Only in one case, Analysis II. of the table, is there sufficient water indicated in the analyses to satisfy all of the ferric oxide according to the formula  $2\text{Fe}_2\text{O}_3, 3\text{H}_2\text{O}$ .



rence in the fresh and in the altered rock can best be described separately.

*Occurrence in the Fresh Rock.*—Abundant opportunity is afforded for studying the mode of occurrence of the ocher in the fresh rock over many parts of the area, from the good natural exposures of the rock. (See Figs. 4, 5 and 6.) For the study of the ocher in its relations to the fresh rock, the best section is at the wooden-bridge over the Etowah river, two miles southeast of Cartersville, where the river has cut across one of the quartzite ridges and where extensive mining has been done by the Georgia Peruvian Ocher Company. Here the quartzite has been extensively crushed and shattered from compression, so that it is difficult to determine the original bedding of the rock (Fig. 4).

Concerning the occurrence of the ocher at this locality, Doctor Hayes says:<sup>4</sup> "The ocher forms a series of extremely irregular branching veins, which intersect this shattered quartzite without any apparent system. They frequently expand into bodies of considerable size; and when the ocher is removed, rooms from 6 to 10 ft. in diameter are sometimes left, connected by narrow winding passages. The mining of the ocher has left the point of the ridge completely honey-combed with these irregular passages and rooms.

"The contact between the ocher and the inclosing quartzite is never sharp and distinct, but always shows a more or less gradual transition from the hard vitreous quartzite, to the soft ore which may be easily crushed between the fingers. The quartzite first becomes stained a light yellow, and loses its compact, close-grained texture. This phase passes into a second, in which the rock is perceptibly porous, having a rough fracture and a harsh 'feel,' and containing enough ocher to soil the fingers. In the next phase the ocher preponderates, but is held together by a more or less continuous skeleton of silica, although it can be readily removed with a pick. The final stage in the transition is the soft yellow-ocher, filling the veins, which crumbles on drying, and contains only a small proportion of silica in the form of sand-grains.

"The intermediate zone between the pure ocher and the

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<sup>4</sup> *Trans.*, xxx., 415-416.

quartzite is usually a few inches in thickness, although it may be several feet between the extremes, and, on the other hand, sometimes only a fraction of an inch."

Microscopic study of a large number of thin sections of the ocher-charged quartzite collected from all parts of the area discloses, with but few exceptions, either the former or the existing presence of pyrite. In many of the sections at least a part of the pyrite is entirely fresh and unaltered, but in a majority of them the pyrite has been completely oxidized, leaving the original space occupied by the mineral only partially filled, as a rule, by its alteration-product, iron oxide. Between these two extremes, of scant iron oxide partially lining the cavity and pseudomorphic limonite filling the entire cavity, all gradations are traced.

The full and accurate description quoted above from Hayes of the ocher-occurrence in the fresh quartzite in the section exposed at the wooden-bridge over the Etowah river, similarly applies to the remaining exposures over the area studied by me, in which sections show the occurrence of ocher in the fresh rock.

*Occurrence in the Residual Clays.*—The area is one of profound atmospheric decay, and exposures of the fresh rock are rarely seen, except on the steeper ridge slopes and crests. The ocher cuts the enclosing clays in a very irregular manner, forming a series of irregular branching deposits which correspond to veins in the fresh rock. The ore-bodies narrow and widen, thin and thicken, throughout their extent. Irregularity results both as to the vertical and the lateral distribution of the deposits. The contact between the ocher-bodies and the surrounding clays is never entirely sharp, but a more or less gradual transition from the clays to the pure ocher is usually shown. The ocher-charged clays at the point of contact lessen considerably in the ocher-content a short distance away; and ocher is entirely absent from the clays at some distance from the point of contact. As is the case of the ocher-bodies in the fresh rock, the transition zone between the clay and the pure ocher is quite variable, from a few inches or less to as many feet between the extremes.

The field conditions make it entirely plain that the position of the ocher in the clays is in all respects similar to that in the

fresh rock. Evidence pointing to leaching or concentration of these bodies upon weathering of the rock is lacking. Somewhat extensive mining on the properties of the Blue Ridge Ocher Company, the Cherokee Ocher & Barytes Company, and the American Ocher Company, affords the best opportunity for studying the mode of occurrence of the ocher in the residual clays.

*Origin of the Ocher.*

The mode of occurrence of the ocher when viewed in its relations to the character and structural conditions of the enclosing rock forms the strongest possible argument for the theory favoring its formation from solution. All evidence, both from field- and laboratory-study, goes to prove that the deposition of the ocher has taken place, not by simple filling of cavities and fissures in the rock, but by a molecular replacement of the original rock. The principal evidence in proof of this is summed up as follows :

1. In nearly every instance of exposure of the fresh rock over the ocher-belt the rock is found to be extensively crushed and shattered and cut in all directions with lines of fracture, by compression. This mechanical action was more than probably accompanied by heat, and the zone of crushed rock afforded ready and natural passage-ways for underground circulating waters, both of which conditions were conducive or favorable to increased chemical action.

2. The contact between the ocher and the surrounding quartzite is never sharp and distinct, but is marked by a gradual transition from the hard quartzite to the soft ore. The transition-zone between the hard quartzite and the soft ocher varies from a few inches to several feet in thickness between the two extremes.

3. The exceedingly irregular character of the ore-bodies and their distribution in the enclosing rock furnish further evidence. The shattered rock is cut in all directions by exceedingly irregular branching veins of the ocher, which narrow and widen, thin and thicken indiscriminately, without apparent regard to system or uniformity.

4. Microscopic study of the transition-portion of the rock makes plain the relations of the ocher to the quartz-rock and the mode of ore-occurrence in the rock. The iron oxide

(ocher) usually cuts across the quartz-grains instead of lying between them. This replacement process is never complete, but all the ocher contains more or less free quartz-grains distributed through it. The grains are very irregular in outline, which distinguishes them from ordinary quartz- or sand-grains, and they were evidently formed from solution.

5. Finally, the replacement-hypothesis is strengthened by the general appearance of the rock which affords in places evidence of solution and redeposition of the silica. Small cavities are frequently observed penetrating the quartzite, and lined with very small crystals of quartz deposited from solution. Also, the skeleton of silica holding the ocher together and the sand-grains disseminated through the purer beds of soft ocher appear to have been derived by deposition from solution, rather than to be composed of grains of the original rock.

The source of the iron oxide, and the solution of the quartz of the rock and its replacement by the iron oxide (ocher) have been previously discussed in some detail by Hayes,<sup>5</sup> and need not be repeated here. After a microscopic study of a large number of thin sections of the ocher-charged and the ocher-free quartzite, I feel reasonably certain that an additional source of the iron oxide not considered by Hayes, is derived from the oxidation of pyrite disseminated through the quartzite. The source of iron oxide from this direction may possibly have been of only secondary importance.

## ECONOMIC FEATURES.

### *Methods of Mining.*

The ocher-deposits to be mined in the Cartersville district form extremely irregular branching veins, which intersect the rock in almost every direction. The ore-bodies may occur enclosed in the hard and fresh quartzite, or they may be entirely enclosed in the residual decay derived from the quartzite. The bodies of pure ocher are usually soft and clay-like in character, and the ore is easily mined with the pick and shovel. They are generally exposed along the slopes and summits of the quartzite hills and ridges.

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<sup>5</sup> *Trans.*, xxx., 403.

On those properties in the district where systematic mining has been done, the method employed consists of tunnels driven into the ridge, from which drifts are worked at suitable points. In this way a number of levels have been worked, one above the other, on several of the properties. Occurrence of the ocher in the fresh rock, as on the Georgia Peruvian Ocher Company's property at the wooden-bridge across the Etowah river, at times necessitates blasting. (See Fig. 4.)

Timbering is necessary in the tunneling, as caving is apt to occur. The underground-mining is also extensive enough to necessitate tramways and lights. The tram-cars are drawn in and out of the mines either by mules or by means of steam and cable. Both are in use in the Cartersville mines.

### *Preparation of the Ocher.*

The only preparation necessary involves the separation of the ocher from its mechanically admixed impurities as mined, which consist principally of clay, sand, and manganese oxide. These are freed from the ocher by a process of washing in running water, floating, and settling of the ocher in vats, from which the water is evaporated.

With one exception, the form of washer in use at the different plants in the district is of the kind used in washing the manganese and brown iron-ores, known as the log-washer, and described by me in a former paper,<sup>6</sup> entitled "Geological Relations of the Manganese Ore-Deposits of Georgia."

The form of washer in use at the Blue Ridge Ocher Company's plant was planned for the purpose of diminishing the grinding and rubbing action of the log-washer, and thereby decreasing proportionately the resulting percentage of finely divided impurities, principally sand and manganese dioxide, which would be floated with the ocher. The washer consists of a V-shaped box about 7 ft. in length, 5 ft. high, and 3 ft. wide at the top. In the bottom of the box is fastened a 3-in. pipe with  $\frac{1}{8}$ -in. perforations along the top at intervals of about 1.25 in. The water is introduced into this pipe, and passes into the box through the perforations in the pipe. Over this pipe revolves a shaft set with 1-in. pins so arranged that those of one row fall

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<sup>6</sup> *Trans.*, xxxiv., 207.

just halfway between those of the next, giving a spacing of 2 in. between pins, if they were aligned in the same row.

By this arrangement the ocher is sufficiently disintegrated without being subjected to the grinding and rubbing action of the ordinary log-washer. The water containing the suspended particles of ocher overflows at the top of the washer-box into a line of troughs, through which it passes into the settling-vats. The ocher is further purified by the settling of a portion of the impurities along the bottom of the troughs in transit to the vats. After the overflow becomes thin, showing that the ocher in the charge is about exhausted, a long narrow door at the bottom of the washer is raised and the sand, clay, etc., mixed with some ocher, are washed through and carried off in a trough as waste.

I am reliably informed by the manager of this plant that the washer of the size above stated easily handles from 25 to 30 tons of ocher per day of ten hours, and that all the necessary work, including cleaning of the troughs, etc., is readily performed by three ordinary laborers.

The ocher passes from the washer, through a line of troughs, into a series of settling-vats dug in the ground, and is exposed throughout the day to the direct heat of the sun. After it has settled, and as much as possible of the water siphoned off by rubber-hose, the remainder of the water is expelled by evaporation. As soon as it is stiff enough to be handled, the ocher is removed from the vats and placed on drying-racks under a shed where the drying-process is completed. (See Fig. 7.) The time required for the evaporation of the water from the ocher in the vats sufficient to admit of handling will average about ten days in clear summer weather. It requires from eight to twelve days of similar weather to complete the drying on the racks before the ocher can be pulverized.

Evaporation is either by natural means—exposure to the heat of the sun—or by artificial means promoted by steam-heating. Several of the plants in the district are fully equipped for both natural and artificial evaporation. The artificial drying is in vats or tanks, arranged in series, in which iron-pipes are run at close intervals along the sides and bottoms for steam-heating. Drying by this method requires usually not longer than one or two days, when the ocher is ready to be removed to the

racks and the drying continued for the usual time, from eight to twelve days, before it is dry enough to be pulverized. While the time is much shortened by the steam-drying from that of the natural evaporation by the sun, the ocher is less desirable than the sun-dried material; the reason for this being that near the pipes the heat is strong enough to dehydrate partially or calcine the ocher, changing its color from yellow to dark red. For this reason some plants in the district have not included an equipment for artificial drying.

After being thoroughly dried, the ocher is pulverized and packed under steam-pressure in barrels and bags of uniform size, ready for shipment.

Fig. 8 presents a view of the plant of the American Ocher Company, two-and-a-half miles northeast of Cartersville. The drying-racks are shown in the rear of the main building.

#### *Uses.*

The principal use made of the Cartersville ocher at present is in the manufacture of linoleum and oil-cloths. For this consumption the principal markets are in England and Scotland, to which the bulk of the Cartersville product is exported. Some of it is used in the United States for a similar purpose. It is also used to a limited extent in the manufacture of paints. By calcining, the ocher is converted into a desirable dark-red pigment.

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### Notes on Contact-Metamorphic Deposits in the Sierra Nevada Mountains.

BY H. W. TURNER, SAN FRANCISCO, CAL.

(New York Meeting, October, 1903.)

IN addition to the deposits mentioned by Mr. Lindgren,<sup>1</sup> the following may be noted:

1. In the "Dardanelles quadrangle," of the map of the U. S. Geological Survey, 3 miles NW. of Tower Peak, there is a considerable mass of quartzite, marble, and dark, fine-grained gneiss. At the edge of the coarsely crystalline white marble, having a width of about 400 ft., is a white schistose rock, which

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<sup>1</sup> *Trans.*, xxxi., 230-231.

under the microscope shows pyroxene. Granitoid rocks and quartz-veins cut both the marble and the pyroxene-schist. The granitoid rock is a fresh quartz-diorite containing primary augite and titanite. One of the quartz-veins at the contact of the marble and pyroxenic schist has been prospected, and much green epidote, garnet, blende and some molybdenite have been found in it.

2. In the "Yosemite quadrangle," east of Iron Creek and on the eastern slope of the mountain at an altitude of 9,215 ft., there are numerous streaks and bunches of garnet-pyroxene-epidote rocks, in which open cuts have been made that in some cases show sulphides, notably blende. From one cut issues a sulphurous gas that is doubtless forming from the oxidation of sulphides. Adjoining the schists containing these contact-metamorphic deposits are metamorphic lavas, probably meta-andesites, containing streaks and bunches of magnetite.

3. In the "Yosemite quadrangle," on the southeastern slope of Mt. Hoffmann, several cuts exposing sulphides have been made in a schist lens enclosed in a granitic rock. A shaft also has been sunk on a lode composed of garnet, epidote, quartz, calcite, and a little galena and molybdenite. The lode is at the granitic contact. Not far distant, but not directly connected with this lode, is a mass of marble.

4. In the "Mt. Lyell quadrangle" there are two series of schistose rocks, one composed chiefly of quartzite and slate, which I have called the Tioga formation; and the other, composed chiefly of metamorphic volcanic rocks, which I have called the Dana formation. Most of the metamorphic volcanic rocks of this district, including both lavas and tuffs, are now schists. Iron Mountain is largely made up of these volcanic schists, containing veins of magnetite, which have been exploited to some extent. The property is now owned by the Minarets Mining Co. The magnetite ore shows traces of copper carbonate; but the original copper mineral has not been determined. The magnetite is associated with a green, massive, fibrous hornblende, which contains prisms two or more inches in length of a light-colored mineral, possibly apatite. An analysis of the ore shows from 0.249 to 0.454 per cent phosphorus, but no titanitic acid. Although the magnetite veins are not at the immediate contact of the neighboring granitic rocks,



they should perhaps be classed with contact-metamorphic deposits.

5. In the foot-hills of Madera county, Cal., there is a belt of contact-metamorphic schists (chiefly mica and chialstolite-schists), containing copper-deposits at several places. Two of these deposits, the Ne Plus Ultra, belonging to the California Copper Co., and the Buchanan, were examined. The ore of the Ne Plus Ultra mine is chalcopryrite in a gangue composed largely of a colorless hornblende, resembling tremolite. There are also at this place many dikes of diorite-porphyry. In the clay-schist country-rock there are numerous crystals of chialstolite, often two inches in length. These chialstolite-schists belong to the Mariposa formation. Granitic rocks occur not far distant toward the east. At the Buchanan mine, the enclosing schists contain garnet. The ore is chalcopryrite, associated with a chlorite (resembling delessite), colorless amphibole, pyrite, pyrrhotite, garnet, quartz, and reddish-brown biotite. Granitic rocks are near by.

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### Tombstone and Its Mines.

BY WILLIAM P. BLAKE, F.G.S., DIRECTOR, ARIZONA SCHOOL OF MINES,  
TUCSON, ARIZ.

(New York Meeting, October, 1903.)

IN a former paper read at the Washington meeting of the Institute, February, 1881,<sup>1</sup> I presented a general view of the geology and veins of Tombstone as then developed. Considerable additions have been made to our knowledge of the ore-deposits since that date, nearly all the mines having been worked down to the general water-level of the district. This water was a bar to further progress in depth and costly pumps were erected at the Grand Central mine and at the Contention. By the united action of these pumps the water was gradually lowered, not only in these two mines, but in all the important mines of the district. Pending a consolidation of interests, or an equitable distribution of the cost of unwatering, both pumping-plants were destroyed by fire, and the mines were closed

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<sup>1</sup> "The Geology and Veins of Tombstone, Ariz.," *Trans.*, x., 334-344.

down and remained practically idle for 20 years, until, in 1902, by the efforts of Hon. E. B. Gage, of Arizona, the desired consolidation of the mining-interests was effected and an organization was formed under which the mines have been re-opened, with the intention of working them to a greater depth.

A new and large four-compartment shaft has been sunk to the 600-ft. level of the Contention mine, or just above the water, 569 ft. below the collar of the shaft. At this level powerful pumps have been placed, and sinking has been extended to the 700-ft. level, where other pumps have been installed and will throw water to the surface.

The pumps are from the Prescott Steam Pump Company of Milwaukee. They are of the direct-acting type, triple expansion, with steam cylinders in pairs of 39 in., 23 in. and 15 in. in diameter. The diameter of the water-plunger is 13 in.; stroke, 24 in.

Tombstone is now directly connected by rail with Fairbank, in the San Pedro valley, and with the overland trunk-lines.

Some of the more important developments and additions to our knowledge of the ore-deposits may be briefly enumerated.

1. The West Side vein has been extensively worked and has produced a large amount of ore, not only from the lateral deposits in limestone, known as "flats," or "blanket-deposits," but from ore-bodies on the plane of the vein to a depth of about 500 ft., or to the water-level.

2. It is ascertained that the lateral deposits, flats, or blanket-deposits, extending from the plane of the vein, or lode, and generally into the beds of limestone, follow the crests of the anticlinal folds rather than the synclines or other portions of the beds.

3. Several such ore-laden anticlinal "folds," or "saddles," or "rolls," as they are locally known, have been developed and quantities of valuable ore have been taken from them.

4. The blanket-deposits have great extension compared with their width, and follow the crests of the folds in their downward pitch away from the intersecting lodes. Several such deposits have been followed from the West Side vein nearly to the water-level.

5. The blanket-deposits appear to have been reinforced, or

enriched, by each successive crevice, or vein, crossing them or the limestone anticlines.

6. The former observation and statement, that the heaviest or largest ore-bodies are found on the lower or down-hill side of intersection of the plicated rocks with the crevices or lodes, is sustained and exemplified.

7. Much light has been thrown upon the question of the origin of the manganiferous ores of the Lucky Cuss, the Luck Sure, the Knoxville, and other mines, by the discovery of a mass of manganese sulphide—the mineral species alabandite—in the midst of the limestone upon one of the lower levels of the Lucky Cuss mine, near the contact of the limestone with the granodiorite. There is little reason to doubt that included masses or kidneys of alabandite in the limestone are the source, by decomposition and precipitation, of the manganiferous ores of the pipes or chimneys so common from the surface downwards. These deposits, by the form and relation to the limestone, indicate their deposition from solutions flowing downwards. The alteration of the sulphide by oxidation, with the formation of manganese sulphate solutions, would give the conditions requisite. Such solutions would follow the crevices downwards, enlarging them by solution of the walls, at the same time depositing crusts and masses of manganese oxide by replacement, while calcium sulphate would flow away. However, we find considerable quantities of calcite in association, which is indicative of its formation as one product of the interchange.

This explanation of the origin of the manganese oxide ores also explains the peculiar, irregular forms in which they are found. They occupy irregular crevices without the tabular form of the filling of regular fissures. They are often pipe-like, as shown in sections of the Lucky Cuss and of the Knoxville mines.

The formation of manganese oxides from manganese sulphide, and of oxidized iron-ores from iron sulphide, are analogous.

## A Laboratory Study of the Stages in the Refining of Copper.

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(New York Meeting, October, 1903.)

### INTRODUCTION.

IN refining copper, the metal is melted down in a reverberatory furnace in a more or less oxidizing atmosphere and then further subjected to an oxidizing smelting in order to eliminate the common impurities, most of which have a stronger affinity for oxygen than has copper. In these operations some of the copper is oxidized to cuprous oxide and dissolved by the metal bath. When the quantity of dissolved cuprous oxide has reached about 6 per cent, the metal is said to have been brought to "set-copper." A button-sample will show a depressed surface and, when broken, a single bubble at the apex of the depression; the fracture will be brick-red and dull. It is essential to carry the oxidation to this point in order to know that the impurities have been oxidized as far as it is possible under the given working-conditions. Nearly all the cuprous oxide of the set-copper is now reduced to the metallic state by poling, when "tough-pitch" copper will be obtained. A button-sample will show a flat surface. Upon breaking, it will be found that the former bubble has disappeared and that the fracture has become rose-colored and shows a silky luster. The quantity of cuprous oxide allowed to remain in the copper will vary with the impurities still present in the metal and with the degree of pitch that it is desired to reach. It is essential for the general physical and the mechanical properties of the resulting copper that such impurities as arsenic, antimony, bismuth,<sup>1</sup> lead shall be present in the oxidized state, as they are

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<sup>1</sup> This generally accepted statement by Hampe (*Zeitschrift für Berg-, Hütten- und Salinen-Wesen im Preussischen Staate*, 1874, xxii., 121) is doubted by Roberts-Austen (*Journal of the Society of Chemical Industry*, 1894, xiii., 471) who, in discussing Gowland's paper, "A Japanese Pseudo-Speise (Shiromé)," etc., says, "however small the proportion of bismuth in copper might be, it always remained free, it did not unite with the copper."

then less harmful than when present in the metallic state. Refiners commonly distinguish "ingot- or cake-pitch" and "wire-bar pitch;" copper brought to the former contains more cuprous oxide than the latter. These two pitches are, however, not absolutely fixed; they vary with the practice of the individual refiner and with thickness of the cake or bar that is to be cast:—The thicker the piece, the more oxygen will have to remain in the metal, if a flat surface is to be obtained. A third degree of pitch aimed at is that required by very thin castings, such as electrodes 0.5 in. in thickness. As this pitch lies beyond that of wire-bar copper and differs from it more than to permit its being designated merely a shading, it may be called "plate-pitch."

Little was known of the structural relations that existed between copper and cuprous oxide until 1900, when Heyn<sup>2</sup> published the results of his investigations on "copper and oxygen." He took pure copper wire, cut it into small pieces, made up charges of 450 grams each, added to them varying quantities of pure cuprous oxide, melted the mixtures (excepting sample No. 1) in graphite crucibles lined with porcelain, inserted into the fused charges the protected couple of a Le Chatelier thermo-electric pyrometer and made the cooling-curves. Sample No. 1, copper wire alone, was fused in a graphite crucible without a lining in order that the graphite might have a reducing effect upon the small quantity of oxide present, and the resulting fused copper be as free from it as was possible. The results of his experiments are given in Table I. and are represented graphically by the freezing-point curve in the subjoined diagram (page 677).

It will be seen from Table I. that upon cooling, only the tests Nos. 1, 4 and 5 show a single fall of temperature, which means that they do not pass through a pasty stage, but freeze suddenly. With the other tests there is a gradual freezing from the beginning to the end of the solidification. With all samples, excepting test No. 1, the cooling is accompanied by surfusion, the depressed temperature rising in every instance to the fixed point of 1084° C. This temperature, corresponding to the freezing-points of tests Nos. 4 and 5 containing 3.4 and

<sup>2</sup> *Mittheilungen aus den Königlichen Versuchsanstalten zu Berlin*, 1900, xviii., 315 see also *The Metallgraphist*, 1903, vi., 49.

TABLE I.—*Results of Heyn's Experiments.*

Test No.	Charge.		Crucible, Lined or Not Lined.	Cu <sub>2</sub> O Calcu- lated from O Found by Analy- sis, (a)	Temperature of Solidification.		Observations on the Galvanometer.
	Cu Grams.	Cu <sub>2</sub> O Grams.			Beginning.	Ending.	
1	450	None.	Not lined.	Per Cent. 0.08	1102	.....	{ Decided retarda- tion at 1102° C. Long retardation at 1095° C., slow fall to 1083° C., quick rise to 1085° C., long retarda- tion, slow, followed by quick, fall.
2	450	?	Lined.	1.16	1095	1085	{ Slow fall, begin- ning at 1089° C., retardation at 1086° C., slow fall to 1083° C., sudden rise to 1084° C., long retardation, slow, followed by quick, fall.
3	450	9.0	Lined.	1.75	1089	1084	{ Fall to 1076° C. without any re- tardation, sudden rise to 1084° C., long retardation, slow, followed by quick, fall.
4	450	22.5	Lined.	3.4	.....	1084	{ Fall to 1082° C. without any re- tardation, sudden rise to 1084° C., long retardation, slow, followed by quick, fall.
5	450	18.0	Lined.	3.5	.....	1084	{ Slow fall, begin- ning at 1116° C., very slow fall to 1076° C., sudden rise to 1084° C., long retardation, slow, followed by quick, fall.
6	450	27.0	Lined.	4.7	1116	1084	{ Slow fall, begin- ning at 1149° C., very slow fall to 1077° C., sudden rise to 1084° C., long retardation, slow, followed by quick, fall.
7	450	40.5	Lined.	6.3	1149	1084	{ Slow fall, begin- ning at 1186° C., slow fall to 1079° C., sudden rise to 1084° C., long re- tardation, slow, fol- lowed by quick, fall.
8	450	60.0	Lined.	9.0	1186	1084	{

(a) Reduction by means of hydrogen: "Hampe's Method," *Zeitschrift für Berg-, Hütten- und Salinen-Wesen im Preussischen Staate*, 1873, xxi., 231.

3.5 per cent  $\text{Cu}_2\text{O}$  respectively, which have only one distinct point of solidification, must be the temperature of the eutectic. The V-shaped freezing-point curve of the diagram, plotted from the data in Table I., and the photo-micrographs, Figs. 1 to 8 inclusive, show this fact clearly. Figs. 4 and 5, the eutectic of copper and cuprous oxide with 3.4 and 3.5 per cent  $\text{Cu}_2\text{O}$  respectively, show the characteristic structure, *i.e.*, a conglomerate of the two components arranged more or less in alternate plates which do not cross one another. With alloys of diminishing percentages of cuprous oxide, shown in Figs. 3, 2 and 1, the photo-micrographs give the dark net-work of eutectic and the light mesh-work of copper; with alloys having percentages of cuprous oxide greater than 3.4 or 3.5, the photo-micrographs, Figs. 6 and 7, show patches of cuprous oxide increasing in size embedded in eutectic mixture.

The alloys of copper and cuprous oxide, when in the molten state, form homogeneous solutions similar to salt solutions. Upon the solidification of copper-alloys containing less than 3.45 per cent  $\text{Cu}_2\text{O}$ , the cuprous oxide falls out completely and does not form a solid solution with copper. This is seen clearly in Fig. 1, where the net-work of the eutectic is still visible, when the copper contains only 0.08 per cent  $\text{Cu}_2\text{O}$ . Should any cuprous oxide form a solid solution, its quantity would have to be less than 0.08 per cent. Whether in an alloy with more than 3.45 per cent  $\text{Cu}_2\text{O}$ , the patches of cuprous oxide (Figs. 6 and 7) are pure cuprous oxide, is not settled, but in all probability this is the case.

Finally, Fig. 8 represents the same sample as shown in Fig. 2, *viz.*, copper with 1.16 per cent  $\text{Cu}_2\text{O}$ . In the first case, the specimen has been heated to  $1100^\circ\text{C}$ . and then quenched in water of  $10^\circ\text{C}$ .; in the second, it has been allowed to cool slowly. The disconnected points in the photo-micrograph of the quenched specimen show how the free development of the inner structure has been arrested; in contrast, Fig. 2 illustrates clearly how the eutectic, if given sufficient time, forms a connected net-work.

The leading statements of Heyn's paper, including eight of his nine photo-micrographs, have been repeated here, for the reason that they have an important bearing upon the work to be described; in fact, the suggestion in his conclusions that

observation of the microstructure of refined copper might be substituted for the lengthy determination of oxygen was the cause of undertaking the present investigation. This embodies the examination of fractures of samples taken in different stages of refining, the determination of the oxygen-content, the preparation of photo-micrographs, and, lastly, the planimetric measurement of enlarged photo-micrographs, with calculation of the percentage of oxygen.

#### DESCRIPTION OF SAMPLES.

Button-samples for this investigation were kindly furnished by Mr. W. T. Burns, of the Boston & Montana Consolidated Copper and Silver Mining Co., Great Falls, Mont.; by Mr. M. B. Patch, of the Buffalo Smelting Works, of the Calumet & Hecla Mining Co., Buffalo, N. Y.; and by Mr. G. M. Luther, of the Nichols Chemical Co., Laurel Hill, N. Y.

Sample No. 1 of the Boston & Montana Co. represents cathode copper after it has been melted down in the reverberatory furnace and skimmed, but not rabbled; No. 2 was taken after the rabbling had been completed and the stage of set-copper reached; No. 3 is the sample after the poling has been finished, and the copper is ready to be ladled into wire-bars. The tests made at the works give: silver, 0.8 oz. per ton; arsenic and antimony, 0.0035 per cent; conductivity (hard-drawn) 97.5 per cent; tensile strength, 64,200 lb. per sq. in.; elongation, 1 per cent; torsion-twists in 6 in., 89.

Sample No. 0 of the Buffalo Smelting Works represents set-copper; the remaining six of the set, Nos. 1, 2, 3, 4, 5 and 6, were taken at intervals of 15 minutes during the poling-period: No. 1 was cast after the poles had been in the furnace for 15 minutes; No. 6 is finished refined copper brought to a pitch at which ingots or cakes are cast. Samples Nos. A and B, from another charge, represent copper brought to ingot-pitch and wire-bar pitch respectively, special care having been taken to allow the samples to cool slowly.

The sample of the Nichols Chemical Co. represents plate-pitch, that is, the pitch desired for casting thin electrodes.

#### FRACTURES OF SAMPLES.

The fractures reproduced in Figs. 9 to 18, inclusive, were prepared in the usual way. An incision about 0.125 in. in depth



was made across the convex side of a button with a cold-chisel, the button then clamped in a vise with the incision just protruding above the jaws, and given one or more shearing blows with a heavy short-handle hammer. With set-copper, one blow was sufficient to break the specimen in two; the nearer the sample approached tough-pitch copper, the larger was the number of blows required to obtain a fracture.

Several experiments were made to find the best light and time of exposure necessary, the position of the specimen with regard to the light, and the proper magnification to bring out the details in a photograph. A reversible-back Premo camera, 4 by 5 in., was used with sunlight, the specimen having been placed on a white background. Exposures of 40, 80 and 160 seconds were tried, using a No. 32 diaphragm. With negatives half the size, the same size, one and one-half times and twice the size of the original, exposures of from 40 to 80 seconds gave good results. Trying back-light and side-light, it was found that the latter brought out the structure more satisfactorily than the former. A magnification of one and one-half was necessary to show clearly the details in the photograph.

As it is not easy to have constant conditions with sunlight, electric light from a 16-candle-power incandescent lamp was substituted and an enlarging camera of E. & N. T. Anthony<sup>3</sup> used. With the light placed at about 5 in. from the specimen, so as to make an angle of  $45^{\circ}$  with the face, and using the middle diaphragm, an exposure of 6 minutes gave the best negative. With finely-granular fractures, filtering the light through ground glass was an improvement, but the time of exposure had to be prolonged to 10 minutes; with the coarser structures, better results were obtained without the ground glass.

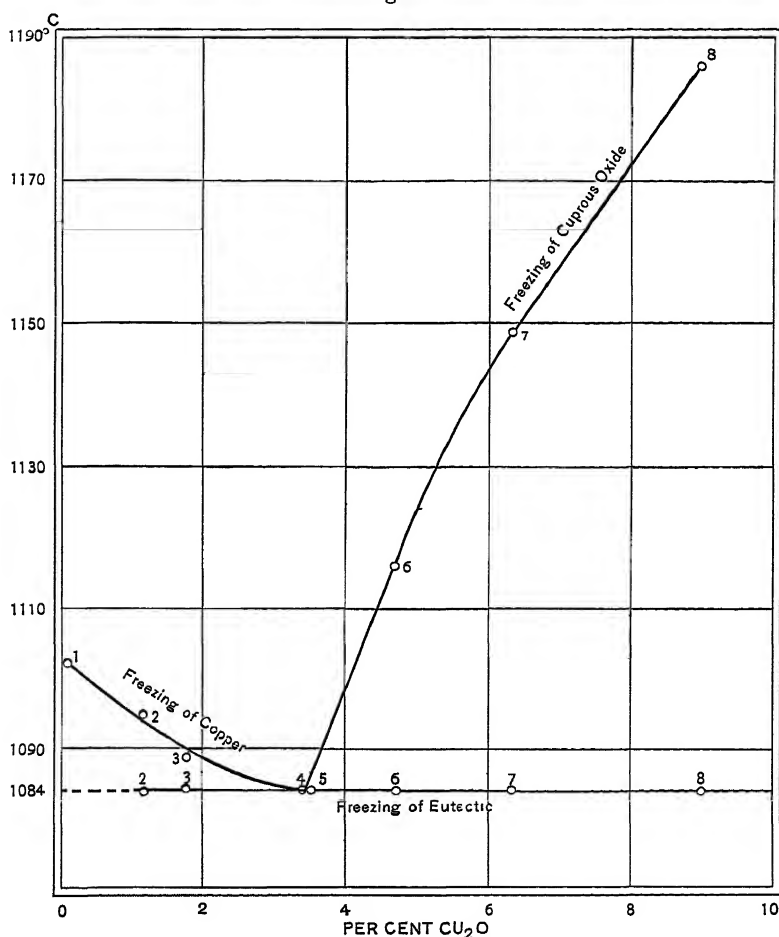
Figs. 9, 10 and 11 give the fractures of the Boston & Montana samples Nos. 1, 2 and 3 in one and one-half times their natural sizes. Fig. 9, cathode copper after melting and skimming, but before rabbling, has a fracture radiated and columnar, luster is absent, the color a dark red. Fig. 10 is set-copper, the fracture has lost its radial character and has become coarse-columnar to coarse-cubical, it remains dull, the color has

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<sup>3</sup> Address, No. 591 Broadway, New York City.

changed to a brick-red; in the apex of the depressed surface there has appeared the characteristic single bubble. Finally, Fig. 11 represents refined copper brought to wire-bar pitch; the fracture is finely-granular and fibrous, the luster is very silky, and the color roseate.

Diagram.



FREEZING-POINT CURVE OF COPPER-CUPROUS OXIDE ALLOYS.

Samples Nos. 0 to 6 of the Calumet & Hecla Co., shown in Figs. 12 to 18, begin with set-copper and end with ingot-copper. The fractures, starting from coarse-columnar and cubical (Fig. 12), lose their columnar character, remaining coarse and cubical (Fig. 13), they become coarsely radiated (Fig. 14), then the

FIG. 1.



FIG. 2.



FIG. 3.

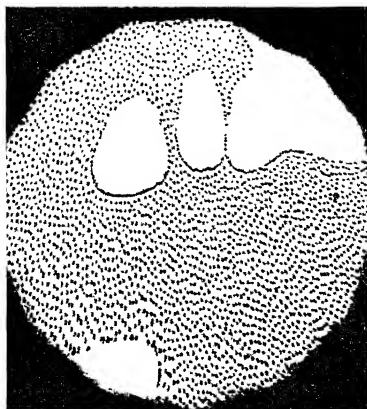


FIG. 4.

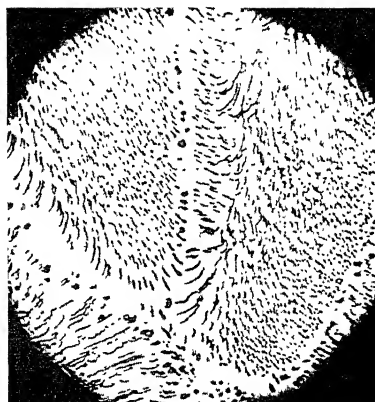


PHOTO-MICROGRAPHS OF COPPER-CUPROUS OXIDE ALLOYS. (HEYN.)  
(Magnified 123 Diameters.)

FIG. 5.

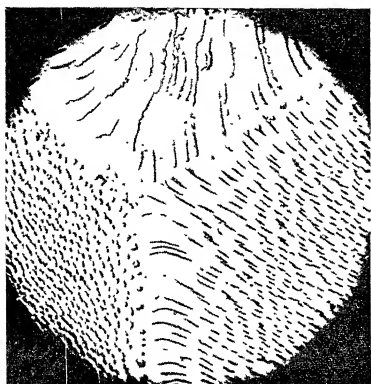


FIG. 6.

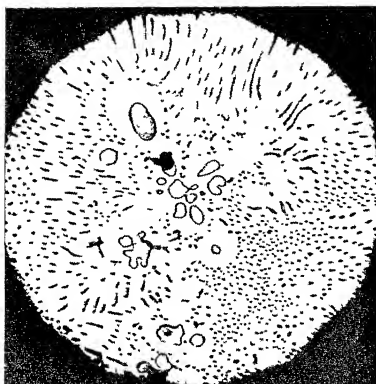


FIG. 7.



FIG. 8.

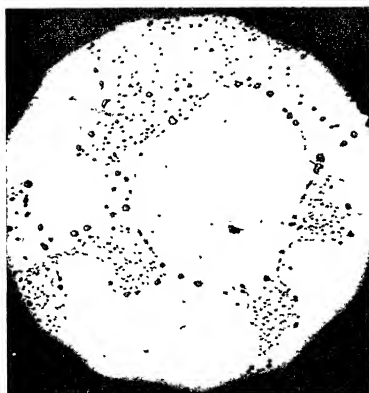
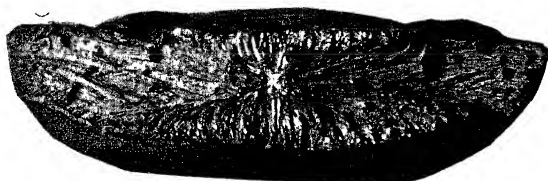


PHOTO-MICROGRAPHS OF COPPER-CUPROUS OXIDE ALLOYS. (HEYN.)  
(Magnified 123 Diameters.)

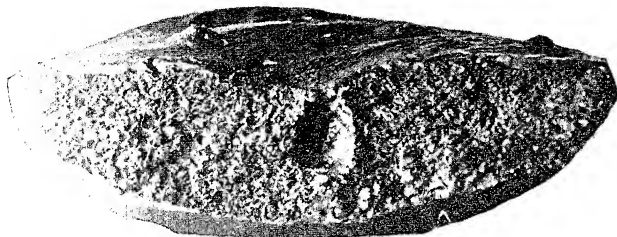
FIG. 9.



Sample No. 1, Boston and Montana Copper. After Melting, contains 3 per cent  $\text{Cu}_2\text{O}$ . (?)

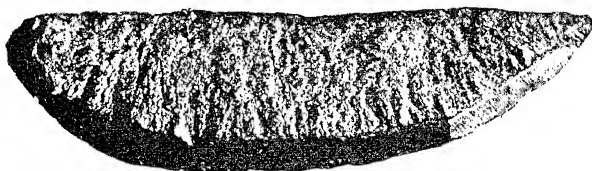
FRACTURE OF COPPER.  
(Magnified 1.5 Diameters.)

FIG. 10.



Sample No. 2, Boston and Montana Copper. Set Copper, contains 6.16 per cent  $\text{Cu}_2\text{O}$ .

FIG. 11.



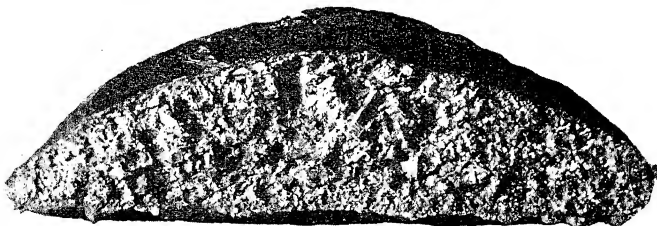
Sample No. 3, Boston and Montana Copper. Wire-Bar, contains 0.51 per cent  $\text{Cu}_2\text{O}$ .

FIG. 12.



Sample No. 0, Calumet and Hecla Copper. Set-Copper, contains 5.76 per cent  $\text{Cu}_2\text{O}$ .

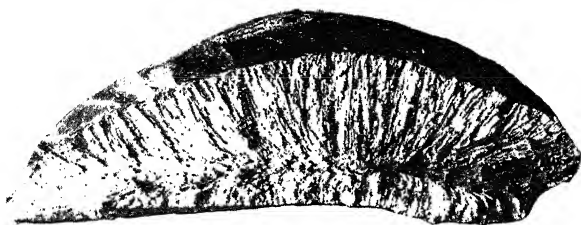
FIG. 13.



Sample No. 1, Calumet and Hecla Copper. After Poling 15 Minutes, contains 5.67 per cent  $\text{Cu}_2\text{O}$ .

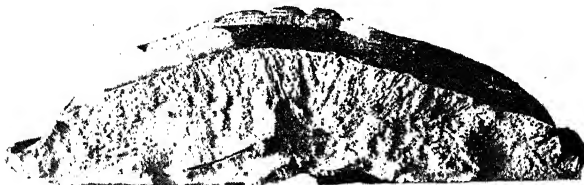
FRACTURES OF COPPER.  
(Magnified 1.5 Diameters.)

FIG. 14.



Sample No. 2, Calumet and Hecla Copper. After Poling 30 Minutes, contains 3.36 per cent  $\text{Cu}_2\text{O}$ .

FIG. 15.



Sample No. 3, Calumet and Hecla Copper. After Poling 45 Minutes, contains 1.86 per cent  $\text{Cu}_2\text{O}$ .

FIG. 16.



Sample No. 4, Calumet and Hecla Copper. After Poling 60 Minutes, contains 1.98 per cent  $\text{Cu}_2\text{O}$ .

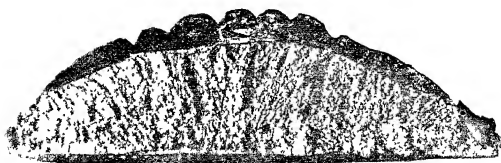
FIG. 17.



Sample No. 5, Calumet and Hecla Copper. After Poling 75 Minutes, contains 0.84 per cent  $\text{Cu}_2\text{O}$ .

FRACTURES OF COPPER.  
(Magnified 1.5 Diameters.)

FIG. 18.



Sample No. 6. Calumet and Hecla Copper After Poling 90 Minutes  
(Ingot Copper), contains 0.47 per cent  $\text{Cu}_2\text{O}$   
FRACTURE OF COPPER.  
(Magnified 1.5 Diameters.)

FIG. 19.

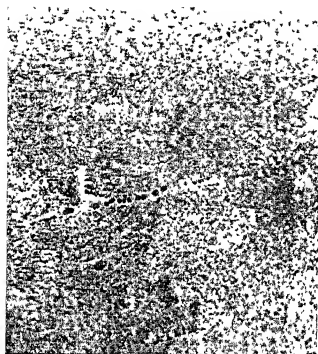


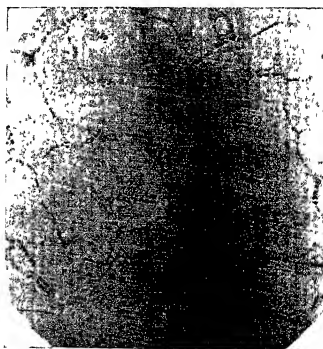
FIG. 20.



Sample No. 1. Boston and Montana Cop-  
per, contains 3 per cent  $\text{Cu}_2\text{O}$ . (?)

Sample No. 2, Boston and Montana Cop-  
per, contains 6.16 per cent  $\text{Cu}_2\text{O}$ .

FIG. 21.



Sample No. 3, Boston and Montana Copper. Wire-Bar, contains 0.51 per  
cent  $\text{Cu}_2\text{O}$ .

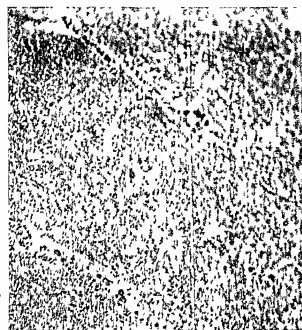
PHOTO-MICROGRAPHS OF COPPER.  
(Magnified 100 Diameters.)

FIG. 22.



Sample No. 0, Calumet and Hecla Set  
Copper, contains 5.76 per cent  $\text{Cu}_2\text{O}$ .

FIG. 24.



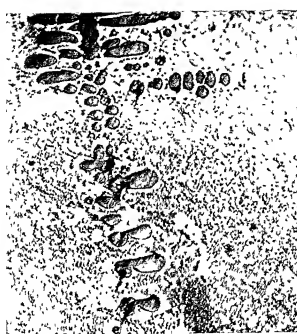
Sample No. 2, Calumet and Hecla Copper.  
After Poling 30 Minutes, contains 3.36 per  
cent  $\text{Cu}_2\text{O}$ .

FIG. 26



Sample No. 4, Calumet and Hecla Cop-  
per. After Poling 60 Minutes, contains 1.98  
per cent  $\text{Cu}_2\text{O}$ .

FIG. 23.



Sample No. 1, Calumet and Hecla Copper.  
After Poling 15 Minutes, contains 5.67 per  
cent  $\text{Cu}_2\text{O}$

FIG. 25.



Sample No. 3, Calumet and Hecla Cop-  
per After Poling 45 Minutes, contains  
1.86 per cent  $\text{Cu}_2\text{O}$ .

FIG. 27

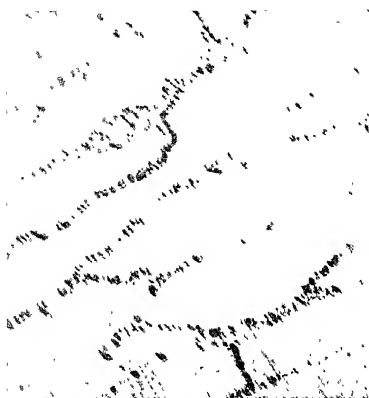


Sample No. 5, Calumet and Hecla Copper.  
After Poling 75 Minutes, contains 0.84 per  
cent  $\text{Cu}_2\text{O}$ .

PHOTO-MICROGRAPHS OF COPPER.  
(Magnified 100 Diameters.)



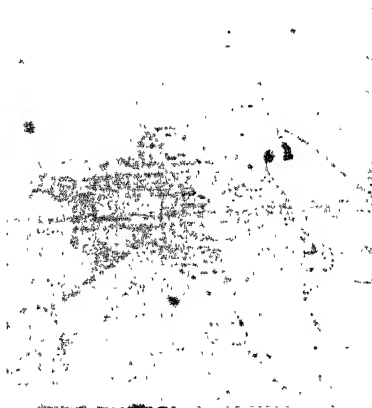
FIG. 28.



Sample No. 6, Calumet and Hecla Copper. After Poling 90 Minutes, contains 0.47 per cent  $\text{Cu}_2\text{O}$ .

(Ingot Copper Cooled Quickly.)

FIG. 29.



Sample "A," Calumet and Hecla Ingot-Copper Cooled Slowly, contains 0.91 per cent  $\text{Cu}_2\text{O}$ .

FIG. 30.



Sample "B," Calumet and Hecla Wire-Bar Copper Cooled Slowly, contains 0.275 per cent  $\text{Cu}_2\text{O}$ .

FIG. 31.



Sample of Nichols & Co.'s Plate-Copper.

PHOTO-MICROGRAPHS OF COPPER.  
(Magnified 100 Diameters.)

radiation assumes finer forms and granulation puts in an appearance (Fig. 15), granulation predominates over radiation (Fig. 16), both become finer (Fig. 17), until with Fig. 18 radiation has been entirely replaced by granulation. In a similar manner the luster, from being absent with Fig. 12, becomes at first slightly silky; then silkiness increases until full silkiness is reached with Fig. 18. The dark brick-red color of Fig. 12 becomes lighter (Figs. 13, 14 and 15), rose-color begins to be seen (Figs. 15 and 16) until full rose-color is reached with Fig. 18.

The observations on the fractures are brought together in Table II. (on p. 686).

#### DETERMINATION OF OXYGEN IN SAMPLES.

The oxygen of the different samples was determined by means of Hampe's method,<sup>4</sup> which consists in reducing the oxide of finely-divided copper (brought to a bright red heat) in a current of hydrogen, the loss in weight giving a measure for the oxygen-content. Hampe<sup>5</sup> and later Heyn,<sup>6</sup> give evidence that the reduction is complete. The apparatus used and recommended by Hampe was somewhat modified, partly along the lines suggested by Archbutt<sup>7</sup> and partly by changes which suggested themselves during the work. On account of the smallness of the samples, it was necessary to use less material for the analyses than did Hampe, *viz.*, from 10 to 13 grams. The apparatus used consisted of a 16.5-in. Kipp gas-generator (charged with hydrochloric acid and feather zinc), a gas-washing bottle filled two-thirds full with a saturated solution of caustic soda, a drying-tower with sticks of caustic soda, a U-tube filled with calcium chloride, a bulb-tube filled with copper borings and a second U-tube filled with calcium chloride.

The bulb-tube, of Bohemian glass,  $\frac{3}{16}$  in. thick, was 8 in. long and had a bulb 3 in. long and 1.25 in. in diameter. It was supported by a frame (5 in. long by 3 in. wide by 2.5 in. high) of "uralite" (an asbestos boarding) having slots 1.25 in.

<sup>4</sup> *Zeitschrift für Berg-, Hütten- und Salinen-Wesen im Preussischen Staate*, 1873, xxi., 231.

<sup>5</sup> *Op. cit.*, p. 234.

<sup>6</sup> *Zeitschrift des Vereins deutscher Ingenieure*, 1900, xliv., 509; see also *Metallographist*, 1900, vi., 48.

<sup>7</sup> *Analyst*, 1900, xxv., 253.

TABLE II.—*Observations on Fractures of Sample-Buttons.*

Properties.	Boston and Montana.			Calumet and Hecla.						
	Samples			Samples						
	No. 1.	No. 2.	No. 3.	No. 0.	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.	No. 6.
Texture.....	Radiated, columnar.	Columnar, cubical, coarse.	Fine granular, fibrous.	Cubical, columnar, quite coarse.	Cubical, coarse.	Radiated, coarse.	Finely - radiated, somewhat granular.	Granular to finely radiated.	Finely granular, with some fine radiation.	Finely granular.
Luster.....	Dull.	Dull.	Very silky.	Dull.	Dull, with specks of silkiness.	Dull, with increasing silkiness.	Light brick.	Light brick to rose.	Half dull, half silky.	Nearly full silkiness.
Color.....	Dark brick.	Brick.	Rose.	Brick.	Brick.	Light brick.	Light brick.	Light brick to rose.	Light brick to rose.	Rose.

deep in the ends to receive the cylindrical ends of the bulb. The frame was placed on a thin sheet of asbestos-paper<sup>8</sup> resting upon a ring-stand. Under this was placed a Tirrel burner with a flame spreader. The whole was enclosed by a frame of heavy asbestos matting (7 in. long by 6 in. wide by 13 in. high) with slots in the sides to receive the protruding cylindrical ends of the bulb.

The mode of procedure was as follows: From 20 to 25 grams of borings<sup>9</sup> were taken for a sample, small bits of iron were removed by a magnet, the borings were washed in a beaker four or five times with alcohol and dried to remove the last traces of alcohol, care being taken to avoid any oxidation of the copper. The borings were then divided into approximately equal parts, transferred to the weighed bulb-tubes and weighed. A bulb was placed in the furnace, connected by rubber tubing with the train of hydrogen-apparatus, hydrogen passed through for five minutes at the rate of six bubbles per second, the gas issuing from the second calcium-chloride tube ignited and the bulb slowly brought to a bright-red heat. When water ceased to appear in the glass leading to the second calcium-chloride tube, the gas-current was reduced so that only three bubbles passed the wash-bottle per second, and the bulb kept at a bright-red heat for one-and-a-half hours. At the end of this time, the supply of gas was again increased to six bubbles per second, the lamp removed and the copper allowed to cool. When cool, the bulb was disconnected, air aspirated through it, and the bulb cleaned and weighed.

The results obtained are given in Table III. It will be noticed that the average percentage of cuprous oxide of the Boston & Montana wire-bar copper is higher than that of the Calumet & Hecla cake-copper, although the former had been brought to a higher pitch and should, therefore, contain less oxygen. The discrepancy may be explained by the fact that the Boston & Montana copper contains more impurities than the Calumet & Hecla; and these impurities are present as oxides.

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<sup>8</sup> Asbestos was chosen after sheet-iron (used by Archbutt) had failed with the heating apparatus used, and copper had become oxidized and perforated too quickly.

<sup>9</sup> In boring, the drill was run very slowly to prevent the heating of the sample.

TABLE III.—*Determinations of Oxygen in Samples of Copper.*

Sample.		Weight of Copper Used.	Weight of Oxygen Found.	Quantity of Oxygen in Copper.	Quantity of Cuprous Oxide in Copper.
		Grams.	Grams.	Per Cent.	Per Cent.
Boston & Montana..	No. 1 {	9.9251	0.0319	0.322*	2.88*
		10.2990	0.0347	0.337*	3.015*
		10.1953	0.0702	0.688	6.159
	No. 2 {	13.2269	0.0873	0.659	5.905
		6.7495	0.0054	0.056	0.503
	No. 3 {	6.7671	0.0048	0.058	0.523
		10.5543	0.0679	0.643	5.757
	No. 0 {	11.5738	0.0744	0.643	5.755
		12.6041	0.0791	0.628	5.618
	Calumet & Hecla...	No. 1 {	10.9386	0.0700	0.640
11.9961			0.0430	0.358	3.207
No. 2 {		12.1626	0.0456	0.375	3.360
		11.8279	0.0242	0.205	1.830
No. 3 {		16.2051	0.3440	0.212	1.900
		12.6454	0.2940	0.221	1.980
No. 4 {		13.8294	0.3080	0.223	1.990
		13.3018	0.1250	0.094	0.841
No. 5 {		18.2256	0.1690	0.093	0.839
		11.3544	0.0059	0.052	0.465
No. 6 {	12.2972	0.0072	0.058	0.524	
	13.1625	0.0068	0.052	0.463	

\* These samples contained some sulphur, which was given off as hydrogen sulphide; the figures represent, therefore, oxygen plus sulphur, and not sulphur alone—and the figures in the last column ought to be too high, but, as seen by photo-micrograph, Fig. 19, they are very much too low.

#### MICROSCOPICAL EXAMINATION OF SAMPLES.

The pieces of copper used for making micro-sections were sawed out as nearly as possible from the center of a fracture, as it was thought that some segregation might have taken place in the cooling. Later observations, however, showed that this precaution was unnecessary as long as the superficially oxidized surface was excluded. The samples were all finished with the polishing-machine made by the Boston Testing-Laboratories.<sup>10</sup> In using the machine for rough polishing, the emery on the canvas-wheel pitted the surface to such an extent that it became necessary to file the specimen smooth before proceeding any further. In regular work, therefore, the sawed specimens were first treated with a rough, followed by a smooth, file, and then polished with rouge and water on a revolving wooden disk covered with broadcloth. In polishing, consid-

<sup>10</sup> *Metallographist*, 1901, iv., 275.

erable difficulty was encountered at first, as the polished surfaces, when examined under the microscope, showed disturbing scratches. Polishing by hand with rouge and water on a smooth board, covered first with sheet-rubber and then with chamois, gave a more lustrous surface than when the wheel was used, but at the same time it intensified the scratches. This pointed to the probable presence of coarse particles in the rouge. In order to remove them, about one volume of rouge was stirred up with two volumes of water in a beaker, allowed to settle for about 30 seconds and the suspended matter applied with a brush to the broadcloth disk. The results were satisfactory, and all samples were treated in this manner. When polished, they were cleaned with alcohol and wiped dry with chamois. By thus applying the rouge, running of water onto the machine could be dispensed with, which made the whole operation cleaner. Experiments with decanting the suspended rouge from that which had settled, filtering and then applying the filter-contents to the disk did not work well.

In examining the polished sections with the microscope,<sup>11</sup> magnifications ranging from 30 to 750 diameters were tried. As a high magnification did not bring out the structure more clearly than did one of a smaller diameter, but only narrowed the field of observation, a comparatively low magnification of 100 diameters was chosen. This gave a magnification of about 230 diameters on the photographic plate. To the eye the contrast between the black<sup>12</sup> cuprous oxide in the eutectic (or the bluish-black excess-cuprous oxide in samples containing over 3.45 per cent  $\text{Cu}_2\text{O}$ ) with the red-colored copper was clearly visible, but the photographic plate failed to show it. In order to bring out the structure more clearly, various attempts were made to etch with nitric acid, sulphuric acid, silver nitrate and with the electric current, but they did not improve matters. Heat-tinting did some good, but not enough. Yellow and orange-colored screens were then tried; of these the orange-colored glass proved to be the better, especially when a rapid isochromatic plate particularly sensitive to orange and yellow light was used for photographing. The orange light gave the

<sup>11</sup> Made by C. Reichert, Vienna (Bausch & Lomb Optical Co., Rochester, N. Y.).

<sup>12</sup> To Heyn, *op. cit.*, using chemically pure materials, it appeared bluish-black.

copper a yellowish tint, but had little effect upon the cuprous oxide. The copper alone having an actinic effect upon the photographic plate, it appeared white in the positive, and the cuprous oxide black. The time of exposure giving the best results was found to be two-and-a-half minutes.

The Boston & Montana sample No. 1 (Fig. 19), taken after melting and skimming the cathodes, is seen to contain a slight excess of cuprous oxide over the eutectic, although the analysis gives only 3 per cent cuprous oxide. The black crystals are small, but easily distinguished from the cuprous oxide of the eutectic. Sample No. 2 (Fig. 20), set-copper, contains a large excess of cuprous oxide over the eutectic; it shows fern-like forms which spring up in relief against the eutectic background. The fern-like forms are very unevenly distributed; the eutectic field in some places was free from them, in others it was entirely covered with them. Sample No. 3 (Fig. 21), wire-bar copper, shows an evenly distributed fine net-work of eutectic enclosing large meshes of copper.

In the Calumet & Hecla series, sample No. 0 (Fig. 22), set-copper, shows patches of excess-cuprous oxide in the eutectic. Sample No. 1 (Fig. 23), taken 15 minutes after poling had begun, does not differ much from sample No. 0, proving that the reduction had not proceeded very far. On the whole, both samples resemble very much the set-copper sample (No. 2) of the Boston & Montana, although they do not show the fern-like forms so clearly developed. The eutectic, in most cases, is slightly separated from the patches of the excess-cuprous oxide crystals by a narrow band of copper, and the cuprous oxide in the eutectic seems to have separated somewhat from its copper, thus giving the field a spherulitic appearance.<sup>13</sup> In sample No. 2 (Fig. 24), the third taken, reduction has progressed rapidly, but it still contains a slight excess of cuprous oxide over that of the eutectic mixture. It resembles sample No. 1 of the Boston & Montana series. In sample No. 3 (Fig. 25), the eutectic has been passed, and the excess-copper becomes apparent. Sample No. 4 (Fig. 26) shows that little progress was made in the reduction in the 15 minutes that elapsed between the taking of samples No. 3 and No. 4. An explanation for this is that during this period, 45 minutes after poling had begun, the

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<sup>13</sup> See H. M. Howe, "*Iron, Steel and Other Alloys*," Boston, 1903, p. 41.

poles were withdrawn and new ones put in their places. In sample No. 5 (Fig. 27), the cuprous oxide is very much diminished, the eutectic forms a thin net-work enclosing copper in its meshes. Finally, sample No 6 (Fig. 28), represents refined copper brought to ingot-pitch. The net-work of the eutectic is imperfect and broken, and the dark parts of the eutectic are bunched together and larger than expected. The explanation of this peculiar structure may be found, when Figs. 2 and 8 of Heyn's photo-micrographs are compared, by the supposition that the sample was chilled when still above  $1084^{\circ}$  C., the melting-point of the eutectic, which prevented the eutectic from separating out in the form of a continuous skeleton. In order to find out whether this idea was a correct one, a new sample (A), taken at the ingot-copper stage from another furnace-charge and cooled slowly was obtained and examined. Fig. 29 brings out clearly the difference between quick and slow cooling. Fig. 30, sample (B), represents the same batch of copper when ready to be ladled into wire-bar, the sample having been also cooled slowly. The skeleton here also is seen to be continuous and not broken as in Fig. 28.

Finally, photo-micrograph Fig. 31 represents a sample of copper from the Nichols Chemical Co. which has been brought to plate-pitch, *i.e.*, the poling has been carried further than is the case with the highest degree of wire-bar pitch of the Calumet & Hecla Co.

#### AREA-MEASUREMENTS.

Measurements of areas which gave Sauveur<sup>14</sup> such interesting facts seemed very promising when applied to samples of copper containing less cuprous oxide than the eutectic mixture. Measuring the copper areas and deducting them from the total area would give the eutectic area, and from this the percentage of cuprous oxide could be readily calculated. It would be useless, if not impossible, to measure the areas of cuprous oxide in the eutectic; and on account of the unequal distribution of cuprous oxide in specimens with more cuprous oxide than the eutectic, the data would be misleading.

For the purpose of measurement, enlargements were made of sample No. 3 (wire-bar copper) from the Boston & Montana Co., of samples Nos. 3, 4, 5 and 6 (ingot-copper, chilled) from

<sup>14</sup> *Trans.*, xxvi., 878.



the Calumet & Hecla Co.; also of sample A (ingot-copper, cooled slowly) and sample B (wire-bar copper, cooled slowly) from the Calumet & Hecla Co.; and of the sample of plate-copper from the Nichols Chemical Co.

The enlargements measured 16 by 20 in., giving five times the magnification of the photo-micrographs, or about 1,150 diameters. A circle, 12 in. in diameter, was drawn on an enlargement, divided into four quadrants, and measurements made on each. The Amsler planimeter<sup>15</sup> was the instrument employed. It is commonly used for the measurement of indicator-cards, and is accurate to 0.1 per cent. The copper areas on an enlarged photo-micrograph were outlined with a pencil in order to facilitate measurement. All measurements were carried out in duplicate. It took from 1.5 to 2 hours to measure the copper-areas of a photo-micrograph, the time varying with the clearness with which the edges of the eutectic were defined. Thus, samples No. 5 and especially No. 6 of the Calumet & Hecla series were very difficult to measure. In calculating the percentage of cuprous oxide in the eutectic, the figure 3.45 was chosen, being the average of Heyn's two determinations, 3.4 and 3.5.

The degree of accuracy of the measurements, carried out at least in duplicate, is shown by examples in Table IV., in which A-V represent the copper areas of one sample.

TABLE IV.—*Planimeter Measurements of Copper-Areas.*

Copper-Area.	A.	B.	C.	D.	E.	F.	G.	H.
Square inches {	2.35	1.04	1.61	2.24	0.27	10.17	2.56	8.26
	2.33	1.04	1.61	2.24	0.27	10.13	2.56	8.27
	2.35	.....	.....	.....	.....	10.14	.....	.....
Copper-Area.	I.	J.	K.	L.	M.	N.	O.	P.
Square inches {	0.78	2.83	7.90	4.57	3.20	5.22	3.39	3.14
	0.78	2.83	7.87	4.57	3.20	5.20	3.42	3.12
	.....	.....	7.86	.....	.....	5.20	3.40	3.12
Copper-Area.	Q.	R.	S.	T.	U.	V.	.....	.....
Square inches {	5.47	6.57	4.04	2.21	2.91	2.80	.....	.....
	5.45	6.53	4.01	2.21	2.87	2.80	.....	.....
	.....	6.57	4.00	.....	2.84	.....	.....	.....
	.....	.....	.....	.....	2.84	.....	.....	.....

<sup>15</sup> Made in Switzerland, and sold by the Crosby Steam Gauge & Valve Co., Boston, Mass.

The percentages of cuprous oxide resulting from the measurements in the several quadrants and the averages are given in Table V.

TABLE V.—*Planimeter Measurements of Quadrants of Various Samples.*

Sample.	First Quadrant.	Second Quadrant.	Third Quadrant.	Fourth Quadrant.	Average of Quadrants.
No. 3 (B. & M.)...	0.58	0.52	0.36	0.48	0.49
No. 3 (C. & H.)...	2.34	1.76	1.22	1.84	1.79
No. 4 (C. & H.)....	2.00	1.37	1.95	2.06	1.85
No. 5 (C. & H.)....	1.00	1.15	0.31	1.11	0.89
No. 6 (C. & H.)....	0.39	0.93	0.61	0.30	0.56

It will be seen that the measurements of the quadrants of a sample show some discrepancies, as the copper-islands are not uniformly distributed in the eutectic net-work. An excess of constituent in one quadrant is, however, balanced by a lack in another, giving on the whole a very satisfactory average.

The results obtained by fracturing and by chemical and microscopical analysis are brought together in Table VI.

The features relating to fractures have already been summarized. Comparing the cuprous-oxide content obtained by chemical analysis and by planimetric measurement, it will be seen that the percentage of cuprous oxide found by analysis in the Boston & Montana sample No. 3, and in the Calumet & Hecla samples Nos. 3 and 4, is somewhat higher than that by measurement. This may be due to the fact that the chemical analysis gives the total oxygen, that of the copper as well as that of the impurities, while measurement gives only the oxygen of the copper. That the oxygen found by analysis in the Calumet & Hecla samples Nos. 5 and 6 is lower than that obtained by measurement is probably due to the segregation of the cuprous oxide in the eutectic, causing the latter to spread somewhat. Taking the results as a whole, they show that area-measurements of enlarged photo-micrographs of pure coppers containing less oxygen than the eutectic give good valuations of the oxygen-content. Further, it seems entirely feasible to make quickly a close estimation of the percentage of cuprous oxide contained in a sample of copper by simply examining a polished surface with the microscope, when once some experience has

TABLE VI.—Physical Characteristics and Corresponding Content of Cuprous Oxide and Oxygen in Various Samples.

[illegible]

been gained. The mode of operating might be as follows:—To take a button-sample, cool it slowly and quench it when it had solidified, cut out a piece with a circular saw, grind it smooth on a number of revolving wooden disks covered with emery-cloth or on revolving files, polish with rouge and water on a revolving disk covered with broadcloth (a mirror-like surface would not be necessary), and estimate with the microscope the percentage of cuprous oxide present. The whole operation could be done in from 6 to 8 minutes. The poling could then be controlled by the microscope, and the degree of pitch desired for ingot-, large or small cake-, wire-bar- or electrode-copper defined by a readily ascertainable amount of cuprous oxide that should be present.

In conclusion we wish to thank Professors Richards and Fay for many valuable suggestions made during the course of the investigation.

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### Biographical Notice of Theodore D. Rand.

BY THOMAS M. DROWN, SOUTH BETHLEHEM, PA.

(New York Meeting, October, 1903.)

THEODORE DEHON RAND was appointed Treasurer of the American Institute of Mining Engineers by the Council, at the Boston meeting of February, 1873, to fill a vacancy caused by the resignation of the treasurer who had been elected at the first meeting in Wilkes-Barre in May, 1871. He was elected by the Institute to this office in the following May, and yearly thereafter until 1903. He often alluded with amusement to the abrupt way in which he was injected into this position, by my coming to his office and asking him to accept the appointment as Treasurer, on the ground that he was eminently fitted for the position by reason of his legal and business knowledge, his interest in geology and mineralogy, and more particularly because he was a neighbor of the person (to wit, myself) who was slated for the Secretaryship, and they could thus conveniently confer on Institute matters. He consented, on condition that the duties should not involve too much additional work for him. This assurance was easily given, for the Institute then

had less than 300 members, and the Treasurer's work was light. We little expected at that time a membership of more than 3,500, the number at the time of Mr. Rand's resignation, a few weeks before his death. Yet he did not decline the burden on account of its increasing weight. His work for the Institute for more than thirty years was a remarkable record of efficient, faithful, gratuitous service, involving great responsibility and intelligent management. The following minute, adopted by the Council, and entered upon its records, expresses the sincere feeling of all our members :

"Through the death, on April 24, 1903, at Radnor, Pa., of Mr. Theodore D. Rand, for thirty-one years the Treasurer of the Institute, it has lost the most devoted and disinterested of the many members to whose cordial co-operation and personal services its signal success has been due.

"Mr. Rand's professional ability, scientific accomplishments, sterling character and genial personality will be appropriately recognized elsewhere. It is the special purpose of the Council at this time to place upon record a statement of his long and unselfish service to the Institute, and a declaration of the affectionate gratitude of its officers and members.

"Mr. Rand was elected Treasurer in May, 1873, and retained that office by successive unanimous annual re-elections until April 17, 1903, when his resignation, tendered upon the ground of failing health, was regretfully accepted by the Council.

"At the time of his first election, the Institute comprised about 240 members, and the care of its funds involved comparatively little responsibility. But this responsibility was rapidly increased by the growth of its membership and consequent income; and during the period of his service as Treasurer, a sum of more than half a million dollars passed through his hands, without a single instance of error in his accounts.

"For this long service and great responsibility, Mr. Rand received no remuneration, beyond the consciousness that he was thus contributing essentially to the success of the Institute, in which he was deeply interested. This interest was further shown in his personal attendance at the meetings and participation in the excursions of the Institute; and, now that his connection with it has been terminated by death, the Council, speaking, as it does not doubt, for every one of some 3,500 members and associates, regards the frank and grateful acknowledgment of his memorable services as not only the privilege of personal affection, but also the dictate of simple justice."

Service of this kind was characteristic of the man, and was performed for many other societies and organizations, with which he was connected, in the same faithful, accurate and judicious way. Moreover, it was always a willing, cheerful service, bringing, perhaps, even more pleasure and satisfaction to him than to those for whom it was rendered without other reward. In the trivial affairs of every-day life, the same spirit

was shown. With regard to things small, as well as great, the remark once made by a friend held true: "If Rand undertakes it, it will be well and thoroughly done!"

Mr. Rand was born in Philadelphia, September 16, 1837, and died at Radnor, Pa., April 24, 1903. Between these dates is comprised an active, useful and noble life, of which this brief notice can give only a picture, rather than a detailed narrative.

His early education was obtained in Philadelphia, at the Episcopal Academy, and subsequently at the Polytechnic College, a technical school of high standing in its day. It was doubtless at this school that he developed the taste, and laid the foundation of the knowledge, in the natural sciences which he afterwards so signally exhibited. Yet science was not his vocation, but his "avocation;" for he applied himself to the study of law, in the office of the Philadelphia firm of Bullitt and Fairthorne. The circumstance that, after his admission to the bar, the name of the firm was changed, first to Fairthorne & Rand, and later to Rand & Patton, is evidence of his steadily progressive success as a lawyer. He was a forcible and persuasive speaker, and fertile in suggestions for the settlement of differences, and the reconciliation of opposing interests.

It was only a few years before his death that he definitely withdrew from legal practice; yet it was during this busy professional career that he made his numerous valuable contributions to science, especially in the geology and mineralogy of southeastern Pennsylvania. After his retirement, he devoted himself chiefly to his favorite studies, and to his garden at Radnor, near Philadelphia, in which, with great delight, he cultivated rare plants, shrubs and trees.

Mr. Rand's scientific standing was much higher than that of a mere amateur, filling the intervals of other and more serious work with superficial observations in other fields. With botany and chemistry he had much more than a general acquaintance; in geology and mineralogy, his favorite sciences, he was a recognized expert; and in the geology of that part of Pennsylvania which he had specially studied, he fell but little, if at all, short of being the highest authority of his time. He possessed a retentive memory, which responded promptly to demands made upon it; and, in consequence, he was familiarly known among his friends as "the Oracle."

We are all familiar with business men who find in trifling with science their "hobbies" for relaxation and recreation. But one who judged Mr. Rand by his published papers on geology and mineralogy would infer that he was a professor or special student of those sciences, and had devoted to them his exclusive attention.

His love for scientific studies was developed very early. He was one of a group of young Philadelphians, which included Henry Morton, subsequently President of Stevens Institute, Albert R. Leeds, afterwards Professor of Chemistry at Stevens, Robert H. Lamborn, William Trautwine, William J. Palmer and others, who were associated together in the "Philadelphia School of Science," holding their meetings in the rooms of the Philomathean Society, in the old University buildings, on Ninth street in Philadelphia, where the post-office now stands. Each member was expected to bring something of scientific interest to the meetings; and Rand's contributions were so numerous that he quite overawed his associates. On one occasion he brought, in small vials, a collection of all the known compounds of lead, made by himself with very primitive apparatus. His contributions to the mineral collections of the Society were also very liberal; and his discovery, at that time, of a deposit of uranite or autunite, in mica-schist, in a quarry located where Thirty-fourth street now crosses on a bridge over the Pennsylvania Railroad into Fairmount Park, at once made him known to mineralogists throughout this country and Europe. A pleasantry much appreciated by his friends was the saying, in this connection, of one of his associates, that Rand ought to discover a new mineral, so that they might have the opportunity of naming it for him—"Randurite" (Rand, you're right!).

In an appreciative notice of his friend Dr. Henry Morton, written a short time ago by Mr. Rand, he describes how he helped Morton, who had highly-developed mechanical taste and skill, to cast a bronze cannon, in the back yard of the house in which he was then living at 140 South Ninth street. This recalled to me the time when I helped Rand, in the same place, to make cacodyl;—the attraction of the operation for both of us being largely, I think, due to the highly poisonous nature and excessively disagreeable odor of that substance.

Bishop Coleman, of Delaware, a fellow-student, in boyhood days, at the Episcopal Academy, has told me of his long walks in the outskirts of Philadelphia with Rand, who, always provided with a hammer, used to come home heavily loaded with minerals. While he inspired a certain awe among his school-mates by his great interest in natural phenomena, he was as far removed as possible from assuming an air of superiority. Quick to come to conclusions, he was always ready to review them and, if necessary, to revise them. He took few things for granted; but he was always eager to get to what was really involved, by laying bare whatever of nature he could master. And with all this eagerness to get at facts, he was keenly alive to what was inspiring or suggestive in the line of his investigations.

Our Secretary supplies me with a reminiscence of one of the Institute excursions in the West, in which two Rands, both members, and both, alas! now gone from us, took part. One was the head of the Rand Drill Co., and the representative also of the Rand Powder Co. The other was our beloved Treasurer, who, at every stop of the train, was to be seen examining rocks or ore-dumps and clipping off specimens with his hammer. At some remote frontier-station a miner boarded the train, inquiring which of the passengers was "Mr. Rand," and received the reply, "Oh, we have two Rands with us: that one yonder breaks the rocks into pieces, and that other one carries the pieces home!"

Mr. Rand's mature scientific work will be found mainly in the *Proceedings of the Academy of Natural Sciences* in Philadelphia. He was elected a member of the Academy, at the age of twenty, in September, 1857; was a member of its Council from December, 1879, Director of the Mineralogical and Geological Section from 1877, and Curator of the W. S. Vaux collection from 1899 until his death. For many years he was also a member of the Hayden Medal Committee of the Academy. His papers in the *Proceedings* of the Academy, thirty-four in number, cover a period of thirty-two years. Most of them deal with the geology and mineralogy of the region around Philadelphia; and his final contribution to the Academy, "Notes on the Geology of Southeastern Pennsylvania," the most important and comprehensive record of his



field- and laboratory-work, is a worthy monument of the painstaking, conscientious and keen student of nature. He contributed articles also to the Reports of the Second Pennsylvania Geological Survey, to the New York Academy of Sciences, and to the *Journal of the Franklin Institute* of Philadelphia. The last mentioned Society he also served with the devotion which was characteristic of him. He was a member of its Board of Managers from 1874, Vice-President from 1897 until his death, and also a member of its Committee on Publication, and its Chairman for the last five years of his life.

Unlike most geologists, Mr. Rand was not much interested in purely scientific theories. When asked for his views as to the theoretical stratigraphy of the region he knew so well, he would reply with good-humored impatience that he needed all his time to investigate and record facts; and that there were plenty who could theorize without them.

Church work interested him. He was one of the strong supporters of the Church of the Atonement in Philadelphia, of which he was at one time a vestryman and Superintendent of the Sunday School; and subsequently he was one of the most active members in the church of St. Martin's at Radnor, up to the time of his last illness. The "Flower Mission" of Philadelphia commanded his earnest sympathy. Once or twice a week he would send from his country home huge basketfuls of flowers for distribution to the hospitals, rising at five or earlier in the morning, and gathering them himself, to be sure they were ready before he left for his city office. And to this life of service, which could not be hid, there was added a quiet giving of himself and his means which only his immediate family, and the recipients of his charities, knew. He was once overheard by a member of his family to say, "I want to shed all the sunshine on others that I can, as I go along!"

We should naturally expect that a man of this character would not be indifferent to the call made on Pennsylvanians to defend their State at the time of the Confederate advance in 1863. Mr. Rand's name is to be found in the roll of the Landis Battery of Light Artillery which, in conjunction with the 32d Regiment of Pennsylvania Infantry, checked the Confederate advance at Carlisle. An intimate friend<sup>1</sup> tells me

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<sup>1</sup> For this incident in Mr. Rand's life, and for many others, I am indebted to his lifelong friend, Mr. William Trautwine, of Philadelphia.

that, two days after the engagement, he walked over to the field where the Battery was posted and inquired where he could find Rand. Captain Landis pointed across a drenched field, filled with soldiers asleep, to a solitary man in an army overcoat, two sizes too big for him, vigorously chopping fence-rails to build a fire. "That's Rand," said the Captain; "and he's a brave fellow, well baptized with fire!"

At the seventy-fourth meeting of the American Institute of Mining Engineers, held in Atlantic City, N. J., in February, 1898, Mr. Rand's twenty-five years of faithful and unpaid service to the Institute received recognition in a testimonial which was presented to him at the close of a banquet, held at the Hotel St. Charles.<sup>2</sup> The contributors to the testimonial were exclusively those who had joined the Institute before 1880, or had been brought into personal contact with Mr. Rand through their attendance at the meetings of the Institute, or through service upon its Council. At the happy suggestion of a member of the Committee in charge of the matter, the testimonial took the form of a contribution to his mineral cabinet—a superb crystal of Siberian beryl, selected by Mr. George F. Kunz, and pronounced by Mr. Rand to be the finest he had ever seen. And it was my privilege to speak the words of presentation, expressing to the treasurer, whom I had induced to take that office a quarter of a century before, the grateful appreciation of his services by his fellow-members of the Institute.

Mr. Rand's life ended peacefully at his home in Radnor, at the age of sixty-six, with his mind still strong, and his mental vision unclouded. He suffered from no disease, but succumbed gradually to the strain of an unusually active life. He has left us the memory of a singularly simple and straightforward character in which great intellectual power was combined with an ardent love of nature, and a consuming desire to be helpful to his fellow-men.

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<sup>2</sup> See Proceedings of the Atlantic City Meeting, *Trans.*, xxviii., p. xxxii.

## Investigations in Thermal Chemistry Showing Atomic Heat-Valency.

BY HALBERT POWERS GILLETTE, NEW YORK CITY.

(New York Meeting, October, 1908.)

IN every chemical reaction heat is either developed or absorbed, and this plus or minus heat of formation is as definite in quantity as the weights of the reacting elements. In this paper I shall show that each chemical element and each chemical radical has its own definite heat-valency,—a fact which, so far as I know, has hitherto escaped the attention of investigators, but, I feel sure, will be admitted after the demonstration here given, and may lead to some practical conclusions of value.

When Berthelot's "law of maximum work" was found to have so many inexplicable exceptions, the pendulum of a scientific theory swung away from a recognition of the truth underlying that proposition; and this reaction went so far that not only has thermal chemistry been relatively neglected by recent investigators, but it is not uncommon now to find textbooks on physical chemistry which practically ignore all that Thomsen, Berthelot and others have done. To this temporary tendency of thought I attribute the failure to recognize the principles herein announced and explained.

Not long ago, while studying tables of the formation-heat of various salts, I noticed that in this respect all potassium salts in solution differ from the corresponding sodium salts by 9.3 cal.<sup>1</sup> I found also that similar differences exist between other salts, not only when the basic radical is varied, while the acid radical remains constant, but also when the acid radical is the variable. Thus chlorine salts in solution differ from bromine salts in this respect by 10.9 cal. Upon further study, I found that Thomsen had noted similar constant differences, but, so far as I can ascertain, he appears to have attached no great significance thereto.

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<sup>1</sup> One calorie (cal.) being the quantity of heat necessary to raise 1 kg. of water 1° C., and equal to 3.968 British thermal units.

Reasoning from these data, I concluded that such differences could be accounted for only upon the hypothesis that each chemical radical has its own constant heat-valency; that, for example, when  $\text{Na}_2\text{Cl}_2$  forms in solution with the generation of 193.0 cal., the  $\text{Na}_2$  furnishes a definite number of these calories and the  $\text{Cl}_2$  the remainder.

The problem thus presented was, to discover the absolute heat-valency of each chemical element or radical. For this purpose, I selected the three compounds:  $\text{KOH}$ ,  $\text{KBr}$  and  $\text{BrOH}$ . According to Thomsen, their heats of formation in solution are:  $\text{KOH} = 116.5$  cal.;  $\text{KBr} = 90.2$  cal.;  $\text{BrOH} = 26.1$  cal. If the hypothesis that each radical has a constant heat-valency be correct, we may write these heat-equations thus:<sup>2</sup>

$$(1) \quad \text{K} + \text{OH} = 116.5 \text{ cal.}$$

$$(2) \quad \text{Br} + \text{OH} = 26.1 \text{ cal.}$$

$$(3) \quad \text{K} + \text{Br} = 90.2 \text{ cal.}$$

Furthermore, if our hypothesis be correct, we may add or subtract these equations. Subtracting (2) from (1) we have:

$$(4) \quad \text{K} - \text{Br} = 90.4 \text{ cal.}$$

Adding (3) and (4) we have:

$$(5) \quad 2\text{K} = 180.6 \text{ cal.}$$

$$(6) \quad \text{K} = 90.3 \text{ cal.}$$

Subtracting (6) from (3) we have:

$$(7) \quad \text{Br} = 0.1 \text{ cal.}$$

Subtracting (7) from (2) we have:

$$(8) \quad \text{OH} = 26.2 \text{ cal.}$$

Now, according to Thomsen's experiments,  $\text{KCl} = 101.2$  cal.; whence,

$$(9) \quad \text{K} + \text{Cl} = 101.2 \text{ cal.}$$

Subtracting (6) from (9) we have:

$$(10) \quad \text{Cl} = 10.8 \text{ cal.}$$

Proceeding in this manner, I deduced, for all the common chemical elements and radicals in solution, heat-values which I have arranged in tabular form, in Table I.

Upon adding the value of any base radical to that of any acid radical we have the heat of formation of the salt in solution.

Evidently, it does not matter whether I have deduced the

<sup>2</sup> In these formulas the calories are given to the first decimal place only.

absolute value for each radical; for so long as the relative values are correctly stated, the table will give accurate results,—provided the hypothesis upon which it is based be correct. As a matter of fact, I feel sure, for reasons that need not be given here, that the values thus deduced are not absolute. But, on the contrary, I am sure that my hypothesis of definite atomic heat-valency is correct; for the marvelous agreement between Thomsen's experimental values and those given from calculation in Table I., can in no other way be explained.

*Examples.*—To find the formation-heat of  $\text{Na}_2\text{Cl}_2$  in solution, add algebraically to the heat-valency of  $\text{Na}_2$  in Table I. (+171.3

TABLE I.—*Atomic and Radical Heat-Valencies, and Heats of Precipitation.*<sup>3</sup>

		+21.7 $\text{Cl}_2$ .	−0.2 $\text{Br}_2$ .	−30.6 $\text{I}_2$ .	+52.3 $(\text{OH})_2$ .	+41.3 $(\text{NO}_3)_2$ .	+157.8 $\text{SO}_4$ .	−16.2 O.
182.7	$\text{Li}_2$	−16.9	.....	.....	.....	−0.6	−6.7	.....
180.6	$\text{K}_2$	+8.9	+10.2	+10.2	−26.6	+17.0	+6.4	.....
171.3	$\text{Na}_2$	+2.4	+0.4	−2.4	−20.0	+10.1	−0.5	.....
175.0	$\text{Ba}$	−2.1	−5.0	.....	−12.3	+9.4	+5.6	−34.5
173.9	$\text{Sr}$	−11.1	−16.1	.....	−11.6	+4.6	.....	−29.3
165.5	$\text{Ca}$	−17.4	−24.5	−27.7	−2.8	−4.0	−4.4	−18.3
165.2	$\text{Mg}$	−35.9	.....	.....	0.0	.....	−20.3	.....
106.3	$\text{Mn}$	−16.0	.....	.....	.....	.....	−13.8	.....
91.0	$\text{Zn}$	−15.6	−15.0	−11.3	.....	.....	−18.4	.....
78.2	$\text{Fe}$	−17.9	.....	.....	.....	.....	.....	.....
74.6	$\text{Cd}$	−3.0	−0.4	+1.0	.....	.....	−10.7	.....
73.1	$\text{Co}$	−18.3	.....	.....	.....	.....	.....	.....
72.0	$\text{Ni}$	−19.2	.....	.....	.....	.....	.....	.....
59.5	$\text{Sn}$	−0.4	.....	.....	.....	.....	.....	.....
56.9	$\text{H}_2$	0.0	0.0	0.0	+27.4	0.0	−3.8	+27.4
55.3	$\text{Pb}$	+6.8	+10.0	.....	.....	+7.6	.....	.....
55.3	$\text{Ti}_2$	+20.2	−8.3	.....	.....	+19.9	+8.3	+3.1
41.0	$\text{Cu}$	−11.1	.....	.....	.....	.....	−15.8	.....
38.1	$\text{Hg}$	+3.3	.....	.....	.....	.....	.....	.....
5.3	$\text{Ag}_2$	+31.8	.....	.....	.....	+10.9	+4.5	.....

cal.) that of  $\text{Cl}_2$  (+21.7 cal.). The result, 193.0 cal. as the heat of formation of  $\text{Na}_2\text{Cl}_2$ , agrees with Thomsen's value.<sup>4</sup>

The formation-heat of  $\text{Na}_2(\text{OH})_2$  in solution, as determined by adding the tabular values, +171.3 and +52.3, is 223.6 cal., agree again with Thomsen, although not deduced from his data.

<sup>3</sup> In Table I., the calories are given to the second decimal place; hence the occasional difference of 0.1 cal. between the figures in the formulas and in the Table.

<sup>4</sup> While some of these figures must necessarily agree with Thomsen's, since they were deduced, as above shown, from his values, it is quite evident that the great majority are in no wise dependent upon Thomsen's results.

The formation-heat of BaO in solution, obtained by adding the tabular values  $+175.0$  and  $-16.2$ , is  $158.8$  cal., which also agrees with Thomsen's experimental determination.

In this manner I have tested every salt that may be derived from Table I., always reaching a result agreeing, well within the experimental error, with that reported by Thomsen.

It was with great satisfaction that I thus secured a proof of the truth of my hypothesis, that now began to merit the title of theory; but the most satisfactory demonstration of the truth of the hypothesis came later, when I undertook to deduce the heats of precipitation of certain salts. The heat of precipitation is of course equal to the negative heat of solution. Thus the heat of solution of  $\text{Na}_2\text{Cl}_2 = -2.4$  cal., and the heat of precipitation of  $\text{Na}_2\text{Cl}_2 = +2.4$  cal.

In Table I., as we have seen, the heat-valency of each base radical is given directly at the left of the radical, whereas the heat-valency of each acid radical is given just above the radical. The heat of precipitation of each salt is given in the body of the table. For example, the heat of precipitation of  $\text{BaBr}_2$  is  $-5.0$ , which is given in the table directly opposite Ba and directly below  $\text{Br}_2$ .

Certain salts are so insoluble that Thomsen did not find their heats of solution, but these I have deduced. Thus the formation-heat of solid  $\text{Ag}_2\text{Cl}_2 = 58.8$  cal., according to Thomsen, but he did not find the heat of solution of  $\text{Ag}_2\text{Cl}_2$ . This quantity may be deduced from Table I. thus:  $\text{Ag}_2 = +5.3$  cal. and  $\text{Cl}_2 = +21.7$  cal., according to Table I., hence  $\text{Ag}_2\text{Cl}_2$  in solution  $= +5.3 + 21.7 = +27.0$  cal. Subtracting  $27.0$  cal. from the  $58.8$  cal. gives  $+31.8$  cal. as the heat of precipitation of  $\text{Ag}_2\text{Cl}_2$ , hence the heat of solution of  $\text{Ag}_2\text{Cl}_2 = -31.8$  cal., or that of  $\text{AgCl} = -15.9$  cal. In Morgan's book,<sup>5</sup> p. 185, the heat of solution of  $\text{AgCl}$  is deduced from the "formula of increased solubility" at higher temperatures, and by that formula is found to be  $-15.9$  cal., which is an exact agreement with my deduction based upon the hypothesis of atomic-heat valency.

The foregoing is an excellent proof of the truth of my hypothesis, but the following is another and equally good proof:

<sup>5</sup> *Elements of Physical Chemistry*, J. Livingston R. Morgan. New York. John Wiley & Son.

In Table I.,  $H_2 = 56.9$  cal., which was deduced by subtracting  $+ 21.7$  cal. from Thomsen's value of  $H_2Cl_2$  in solution. Since the compounds of  $H_2$  with the acid radicals are liquids, the heat of precipitation for each should be zero; but when we come to  $H_2(OH)_2$ , which is  $2H_2O$ , we find that its value deduced from Table I. is  $56.9 + 52.3 = 109.2$  cal. As a matter of fact Thomsen has found the heat of formation of  $H_2O$  to be  $68.3$  cal., whence  $2H_2O = 2 \times 68.3 = 136.6$  cal., which is just  $27.4$  cal. greater than the  $109.2$  deduced from Table I. This was for a time very puzzling to me until I recalled that the heat of dissociation of  $H_2O$ , into  $H$  and  $OH$ , has been found to be  $-13.7$  cal., whence evidently the heat of association of  $H_2$  and  $(OH)_2$  to produce  $2H_2O$  must be  $2 \times + 13.7 = + 27.4$  cal. Thus what at first seemed to be an inexplicable exception to my theory became in reality the exception that proved the rule. The heat of formation of  $H_2SO_4$ , dilute, as determined from Table I., differs by 1.5 per cent. from Thomsen's experimental value, which I am free to confess is as yet not accounted for, when all other deductions agree far closer with experimental results—seldom differing by more than from 0.1 to 0.2 cal.

TABLE II.—*Heat-Valency of Certain Metals Combining with Chlorine.*

Calories.	Metals.	Calories.
		$+ 21.7$ $Cl_2$ .
137.0	$Al\frac{2}{3}$	$-51.2$
63.6	$Fe\frac{2}{3}$	$-21.1$
56.9	$Sn\frac{1}{2}$	$-0.2$
37.6	$As\frac{2}{3}$	$-11.7$
27.2	$Fe\frac{1}{2}$	$-10.2$
$-3.5$	$Au\frac{1}{2}$	$-3.0$

In Table II. I have given the heat-valency of certain metals that unite in radicals of fractional parts with the acid radicals. Thus  $Al\frac{2}{3} = 137.0$  cal. added to  $Cl_2 = 21.7$  cal. gives  $Al\frac{2}{3}Cl_2 = 158.7$  cal., whence, multiplying by 3, we have  $Al_2Cl_3 = + 476.1$  cal. Gold, it will be noted, has a minus heat-valency with respect to  $Cl_2$ ; but, strange to relate, when we add its value of  $-3.5$  to the  $-0.1$  of  $Br_2$ , we find that we should have given the  $Au$  value a plus sign, if we are to obtain agreement with Thomsen's experimental value. This fact has

FIG. 1.

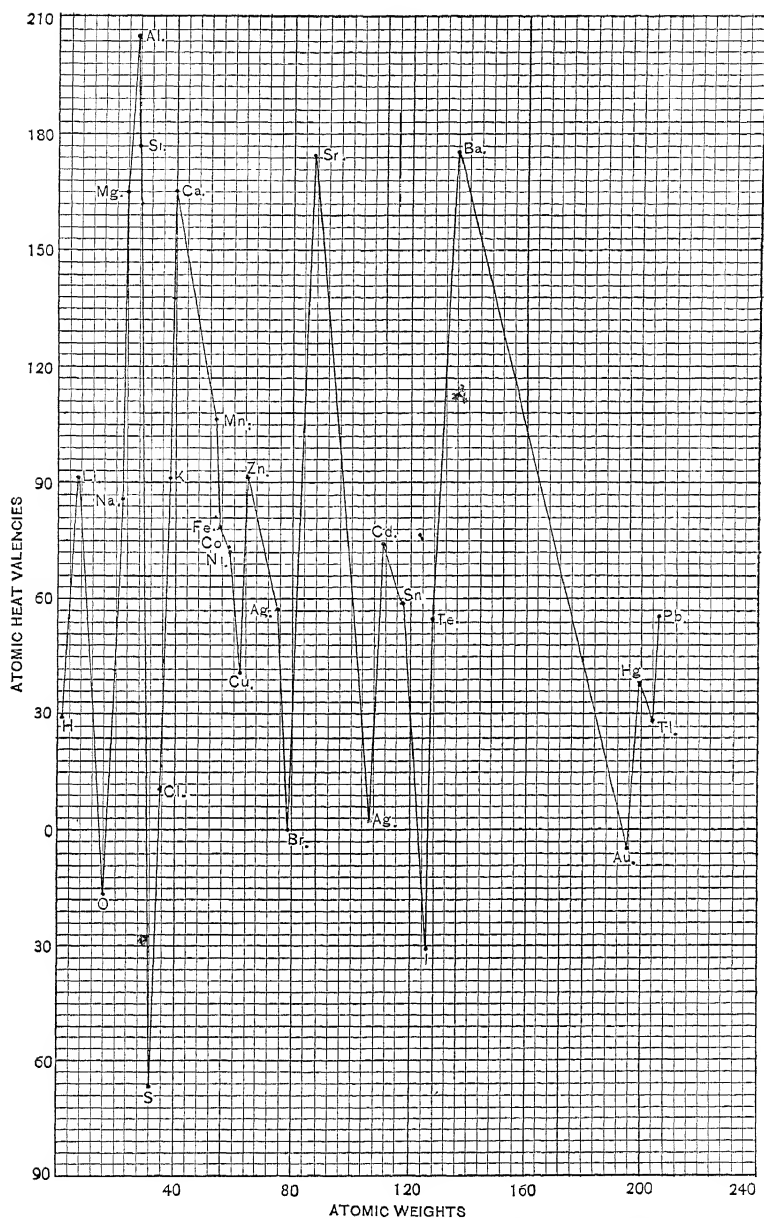


DIAGRAM SHOWING THE RELATION OF HEAT-VALENCIES.

been food for reflection, and I feel quite sure that I have an explanation; but as it would lead into a discussion of what atomic attraction really is, I shall postpone for another paper



any further consideration of this part of the subject. I cannot refrain, however, from calling attention to the fact that the heat-valencies of atoms, as given in Tables I. and II., follow the Periodic Law. To show this I have platted the diagram, Fig. 1. In examining this atomic-heat diagram it is interesting to note that elements which are chemically allied have approximately the same heat-valency; Co and Ni, for example, are almost identical in heat-valency, while Mg, Ca, Sr and Ba vary but slightly. This diagram opens up a most fascinating field of speculation.

Having satisfied myself that each base and each acid radical has a definite heat-valency, I undertook to find some similar law governing heats of precipitation. While this was by no means so easy a task or so satisfactory in results, I can, however, say that in Table III. will be found a remarkably strong parallelism to Tables I. and II. The values in Table III. were derived as follows: According to Thomsen, the heats of solution of  $\text{Cl}_2\text{O}$ ,  $\text{Tl}_2\text{O}$  and  $\text{Tl}_2\text{Cl}_2$  are + 9.4, — 3.0, and — 20.2 cal., respectively. Assuming that each radical has its own constant heat of solution we may write these equations:

$$(11) \quad \text{Cl}_2 + \text{O} = + 9.4 \text{ cal.}$$

$$(12) \quad \text{Tl}_2 + \text{O} = - 3.0 \text{ cal.}$$

$$(13) \quad \text{Tl}_2 + \text{Cl}_2 = - 20.2 \text{ cal.}$$

Solving for  $\text{Cl}_2$  we find its value to be — 3.9 cal. Now, Thomsen has shown that heats of solution are generally some even multiple of 3.2 cal. Therefore, I have called this constant  $m$ , and have assigned to  $\text{Cl}_2$  a value of —  $1m$  cal. in solution. By a process analogous to that used in the construction of Tables I. and II., I have constructed Table III. In Table III., by adding the value of an acid radical to any value below it, we have the heat of solution of the salt whose base radical is directly opposite the lower value taken; thus, the heat of solution of  $\text{Na}_2\text{Cl}_2 = - 1m + (0m + 0.8) = - m + 0.8 = - 2.4 \text{ cal.}$

That in Table III. we are dealing with a law not as simple as in Table I. is shown by the + and — numerals in the body of Table III., and by the necessity of separating the base radicals into two groups. Notwithstanding this lack of simplicity we

TABLE III.—*Heats of Solution.*

	—1m Cl <sub>2</sub> .	—2m Br <sub>2</sub> .	0m SO <sub>4</sub> .	—3m (NO <sub>3</sub> ) <sub>2</sub> .
K <sub>2</sub> ....	—2m + 0.7	—1m — 0.6	—2m + 0.	—2m — 1.0
Na <sub>2</sub> ...	0m + 0.8	2m — 0.4	0m + 0.5	0m — 0.5
Am <sub>2</sub> ...	—1m — 1.4	—1m 0.8	—1m + 0.8	—1m + 0.2
Zn....	+6m + 0.4	+5m + 1.0	+6m + 0.8	.....
Cd....	+2m — 0.2	+2m + 0.4	+2m + 1.8	.....
Mn....	+6m + 0.0	.....	+4m + 1.0	.....
Cu....	+5m — 1.7	+5m + 1.3	+5m — 0.2	.....
	—1m Cl <sub>2</sub> .	0m Br <sub>2</sub> .	—4m SO <sub>4</sub> .	—6m (NO <sub>3</sub> ) <sub>2</sub> .
Pb....	+3m — 0.4	+3m — 0.4	.....	+3m — 1.0
Li <sub>2</sub> ....	+6m + 0.9	.....	+6m — 0.3 (a)	+5m + 0.6
Ba....	+2m — 1.1	—2m — 1.4	+2m + 0.8 (a)	+2m + 0.4
Sr....	+5m — 1.7	+5m + 0.1	+5m — 1.5 (a)	+5m — 1.6
Ca....	+6m + 1.4	+8m — 1.1	+6m — 1.8 (a)	+6m + 0.8
Mg...	+12m + 0.5	.....	+11m — 1.1 (a)	.....

NOTE:  $m = 3.2$  cal. Hence heat of solution of  $\text{CuBr}_2 = (+5m + 1.3) - 2m = 3m + 1.3 = +10.9$  cal.

(a) Calculated.

are at least a step forward in having found that there is some law of solution-heats where formerly all had appeared chaotic. For the present I refrain from going into any explanation of the reasons for these comparatively constant radical heats of solution.

Finally, in closing, a word as to Berthelot's "law of maximum work" may not be entirely out of place. Briefly stated, his law proclaims (1) that no chemical reaction will occur unless heat is given off, and (2) that, as between two possible reactions, that one will occur which involves the maximum production of heat. There is no doubt that the "law" as announced is not a law, but the time will doubtless come when in a modified form that recognizes the "law of mass action" and other laws, a law of maximum work will be formulated. Referring to Table I. we see that if Berthelot's "law" were correct, any metal in the column above  $\text{H}_2$  would displace  $\text{H}_2$  from an acid like  $\text{H}_2\text{Cl}_2$ , or, in other words, would dissolve in the acid; while any metal below  $\text{H}_2$  would not so act. As a matter of fact, this is exactly what occurs. It is, moreover, true that exceedingly few reactions take place between two salts in solution unless heat is evolved. It is not my purpose to defend Ber-

thelot's "law" as it stands, but to emphasize its underlying verity.

In conclusion, I desire to call attention to the dearth of American literature on thermal chemistry and, in fact, on physical chemistry in general. The admirably lucid work of Muir,<sup>4</sup> is the only book in the English language containing much information on the subject. Roberts-Austen's book<sup>5</sup> contains a good chapter on thermal chemistry and a table of heats of formation.

It is to be hoped that American schools of mining engineering will soon see the importance not only of teaching as much of physical chemistry as already is known, but of training students in experimental research-methods with a view to the further development of the science of metallurgy. I am firmly of opinion that further development of the laws of physical chemistry will eventually make of chemistry an engineering science, rescue it from the existing rule-of-thumb methods, and revolutionize electro-chemical and metallurgical practice.

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### Osmosis as a Factor in Ore-Formation.

BY HALBERT POWERS GILLETTE, NEW YORK CITY.

(New York Meeting, October, 1908.)

FROM the known laws of physical chemistry I believe it can be shown that progressive mass movement of water solutions in channels has seldom been the means of ore-concentration in veins. It is my purpose in this paper to show that the force known as "osmosis" has been the principal factor in ore-formation. Convection-currents have doubtless supplemented osmosis, and the two working together have been the agencies that have gathered the scattered particles of rare minerals into the larger masses which are called ore.

Osmosis is commonly thought of as a vague, feeble force

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<sup>4</sup> *Elements of Thermal Chemistry*, M. M. P. Muir. New York. Macmillan & Co. (1885.)

<sup>5</sup> *Introduction to the Study of Metallurgy*, W. C. Roberts-Austen. London. C. Griffin & Co. (1898.)

that causes slow diffusion of dissolving matter through the solvent. That it is a measurable force, often of great intensity, few know, except those who are familiar with the laws of modern physical chemistry. Therefore, I may be pardoned for going briefly into the elementary mathematics of osmotic pressure and of convection-currents.

The velocity of water moving through any channel is given by the Chézy formula:

(1)  $v = c \sqrt{rs}$ , in which  $r$  is the hydraulic mean radius;  $s$ , the slope of the channel in which gravity is the sole propulsive force; and  $c$ , the coefficient of roughness of the sides of the channel.

From this formula it is evident that the smaller the channel, the smaller the velocity of water under a constant head. In rock-crevices, therefore, where the head is constant, water moves much more slowly than in large fissures, and in moving through sand or porous rock it is to an even greater degree retarded. Hence, if a mass of water-saturated sand be heated, the finer the sand-grains, the more slowly will the temperature rise at parts remote from the source of heat. Conversely, cooling is retarded the smaller the interstices or channels through which the convection-currents move. The importance of this fact will appear later. Osmosis is the force which drives a solute through a solution. When a substance dissolves it behaves in many respects exactly as if it were vaporized. In fact, it has been proved by Van't Hoff that "The osmotic pressure of a substance in solution is the same pressure which that substance would exert were it in gaseous form at the same temperature and occupying the same volume."<sup>1</sup>

Expressed mathematically the law is:

$$(2) p = \frac{RT}{V}$$

in which,  $p$  = osmotic pressure in pounds per square inch.

$R$  = the gas constant = 1,206 lb. per sq. in.

$T$  = absolute temperature =  $t^\circ$  (Centigrade) + 273.

$V$  = volume of the solvent containing one molecular weight of the solute.

$$(3) V = \frac{100 M}{r}$$

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<sup>1</sup> *Elements of Physical Chemistry*, by J. Livingston R. Morgan, New York, John Wiley & Son, p. 96.

in which,  $M$  = sum of the atomic weights of the atoms in a molecule of the substance.

$r$  = the rate per cent. of the solution.

Hence:

$$(4) \ p = \frac{1,206 \ r \ (t^{\circ} + 273)}{100 \ M} = \frac{12 \ r \ (t^{\circ} + 273)}{M}.$$

I have deduced equation (4) for convenience in ascertaining and comparing osmotic pressures. Thus, sugar is  $C_{11}H_{22}O_{11}$ . Hence for sugar,  $M = (12 \times 11) + (1 \times 22) + (16 \times 11) = 342$ . For a 2-per-cent. sugar solution,  $r = 2$ . Therefore at  $15^{\circ} \text{C}$ . we have the osmotic pressure according to equation (4) as follows:

$$p = \frac{12 \times 2 \ (15 + 273)}{342} = 20.2 \text{ lb. per sq. in.}$$

By actual experiment  $p$  was found to be 20 lb. per sq. in. under these conditions; and Van't Hoff's law of osmotic pressure has been repeatedly proved by similar tests. From the above statement it is clear that osmosis is far from being a vague or feeble force.

When due to a rise in temperature, or for other reasons, ground-water at any given point begins to dissolve a mineral, that mineral is immediately forced by osmotic pressure out to the extreme confines of the vessel holding the water. Let us suppose, therefore, that a dike-intrusion has so heated the surrounding waters that they have taken up a burden of  $PbS$ ; evidently, then, osmosis will force this  $PbS$  throughout the mass of water into every cranny, crevice, fissure, or cavity. It requires no convection-currents to accomplish this distribution, although convection will hasten the process. Having taken up its full burden of  $PbS$  there comes a period of inactivity, until the temperature begins to fall. This fall of temperature will be most rapid near the surface of the earth and it will be most rapid in the larger water channels, for the reason that circulation there is least retarded.

In the case of dike-intrusion, I believe that the shrinking of the dike-matter will open the largest crevices and channels along the dike itself, and I conceive that surface-waters flowing into these shrinkage-cracks will accelerate the cooling near the dike. Hence, in such cases, contact ore-deposits will be found on one side of a dike, and that side will be the side which originally faced uphill. The down-flowing surface- or ground-water will enter the first crevices it meets and hasten the cool-

ing there, and it is quite evident that where cooling is fastest, crystallization of dissolved mineral will first begin. Let the smallest speck of galena be crystallized out, and immediately osmosis will force more dissolved galena to that point, and thus feed the crystal-mass. Heretofore, the deposition of ore nearest the once-molten dikes has been inexplicable, but in no other place could it have started, and, once started, the tendency toward crystal growth must be due entirely to osmotic pressure.

Let us consider for a moment certain other conditions that accelerate cooling, since it is now evident that much depends upon the first cooling of the solution. At the intersection of two fissures the cooling should be most rapid, for at that point two channels supply convection-currents. Hence the deposition of the most insoluble minerals should occur first at vein-intersections, a circumstance which is often verified in known ore-deposits. Again, in fissure walls that have been striated by slipping past one another, the grooves form channels in which convection-currents move most rapidly, and, as should be expected, the most insoluble minerals are found in the grooves. This, I believe, explains "Clayton's law."<sup>2</sup> Since the richest minerals are ordinarily the most insoluble, I offer this general proposition: "Bonanzas should be sought at points where circulation has been freest, namely, in the widest and most open channels."

Since the relative solubility of minerals has evidently played so important a rôle in ore-formation, a few suggestions may not be out of place. Hot waters carrying metallic sulphides in solution must carry also a great burden of alkaline salts. In fact, the solubility of the metallic sulphides is due to the presence of these alkaline salts which form new ions with the sulphides, just as potassium cyanide dissolves gold salts by forming new ions with the gold. One well-known geologist has erred in assuming that the "solubility product" of every mineral is increased by the presence of other minerals in solution. This, in passing, I would say is not so; for it is only when new ions are formed with the added elements, that solubility is effected, and there are many minerals which do not combine to produce these new ions. The metallic sulphides and the ions formed by them have a very low "solubility product" at best, and when a lowering of temperature causes the

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<sup>2</sup> *Ore-Deposits of the United States and Canada*, J. F. Kemp, p. 49.

sulphides to crystallize out, the "solubility products" of the substances in solution are often so increased that more country-rock is dissolved. In this manner for every particle of sulphide deposited by crystallization, a particle of country-rock may be dissolved. This, I take it, is the true explanation of the so-called "metasomatic replacement" which has been regarded as a true chemical reaction, or precipitation. I have never been able to believe that galena and other natural metallic sulphides are chemical precipitates, and one of my strongest reasons for this disbelief has been the fact that chemical precipitates of lead and similar sulphides are flocculent and never crystalline. How cubical galena could be precipitated chemically from solution, by any reducing agent in nature, has ever been a most serious stumbling block. I am convinced now that practically all of the hypothetical equations written by geologists to account for ore-deposition by chemical reaction are erroneous. Many of these equations can be demonstrated to be false from the laws of thermal chemistry, and few of them appear even plausible when viewed from a thermo-chemical standpoint. It is not now my purpose to tear down, but to build up. I am wholly in accord with Professor Kemp and other geologists who voice the movement against the chemical theories of ore-deposition which have so long occupied the arena. It cannot be denied, however, that very many ore-deposits have come from minerals carried in solution to the chambers where they are now found.

It has been urged in the past that the formation of ore-deposits from hot-water solutions could not have occurred, for two reasons: (1) Because the same heat that would cause the solution of the precious minerals would also cause the solution of the rocks in which those minerals are scattered, and conversely, cooling would precipitate them both together, and, (2) Because hot springs are not now depositing metallic sulphides except in very small amounts and then only in the midst of a mass of gangue.

I trust that I have indicated how osmosis effects the distribution and final concentration of minerals, without any appeal to hot-water-spring theories or to chemical precipitation from cold-water theories, for I am convinced that we must look to the laws of physical chemistry for an explanation of ore-formation, and not to the laws of physics or of chemistry alone.

## A Proposed Filter-Press Slimes Plant.

BY FRANCIS L. BOSQUI, SAN FRANCISCO, CALIFORNIA.

(New York Meeting, October, 1903.)

THE following paper embodies a report which I made on the filter-press treatment of slimes at the Liberty Bell mine, Telluride, Colo. At the time this report was submitted, the management deemed it wiser to defer the installation of so costly a plant as the one required, in the hope that some of the many investigators in the field might soon evolve a cheaper and more perfect method of slimes-treatment, and one better suited to the conditions at Telluride.

The Liberty Bell ore consists essentially of quartz and clay, with iron pyrites disseminated through the quartz in very fine particles. There is considerable iron and manganese stain in some of the ore, the latter especially in the clays, which are sometimes quite black. Occasionally particles of galena, zinc-blende, gray copper and a few other minerals are seen, but they are comparatively rare, and not in sufficient quantity to affect considerations of treatment. Free gold is occasionally seen in the quartz, and one or two instances of wire silver have been observed.

The milling-plant consists of an 80-stamp amalgamating mill which crushes through 20-mesh screens; 20 Wilfley concentrating tables, taking out, in iron sulphides, about 2 per cent. of the total ore crushed; and a cyanide plant consisting of two tiers of 33 ft. vats, five in each tier, fed by revolving pipe-distributers. The slimes overflow from the upper settling tanks through gates with adjustable outlets, and thence to a large canvas-plant, which effects only a small saving. This slimes product, of which about 100 tons escape per day (or 38 per cent. of the total ore crushed), contains about 3 per cent. of dry solids. Of this approximately 93 per cent. will pass a 200-mesh screen. The slimes range in value between \$3.90 and \$4.87 per ton, the values being about equally distributed between gold and silver. The actual averages for ore and slimes during the years 1900-



01, 1901-02, 1902-03—using the market values of silver for the respective years—were as follows :

Year.	Value of Ore Milled Per Ton.	Value of Slimes Per Ton.	Value of Sands Per Ton.
1899-1900.....	\$8.00	\$3.90	\$2.91
1900-1901.....	8.13	4.05	2.78
1901-1902.....	9.55	4.87	3.24

The decantation method of slimes-treatment, as practiced in South Africa, was abandoned after a fair trial, for the following reasons: The impossibility of settling the slimes to a suitable consistency within a reasonable time, even with the aid of lime; the enormous plant made necessary by the retarded settling and the slow dissolution of the silver; and the high consumption of cyanide from the successive dilutions of the strong solution required for the silver. The only alternative was the adoption of the filter-press method. This, however, for so large a capacity, involved a heavy outlay for presses, it having been found necessary to use a primary battery to remove the bulk of water from the slimes.

In the experiments that followed and in the estimates made on cost of treatment and installation, the data and practice of certain filter-press plants now operating in Australia were used as a working basis, modified as the conditions at Telluride required.

To those who have followed the development of filter-pressing in Australia, it is evident that this branch of cyaniding has reached a state of high efficiency there. But the oversanguine metallurgist who is apt to be unduly captivated by the attractions of the new process, and lured by the plausibilities of exploiting concerns who control "improvements" and patents, should be reminded that in Australia filter-pressing was evolved from strikingly unique local conditions—or rather, from a happy conjunction of favorable conditions. I think this may be admitted without depreciating the achievements of those who are responsible for the success of the process. In the Australian ores there is little or no silver to contend with; the slimes are of that permeable sort which admit of rapid operations in the press; the gold exists in a state favorable to

quick and satisfactory extraction, or is rendered so by roasting; and, most important of all, the material is high grade, and can bear a high cost of treatment by a process which happens to be the cheapest and only possible one for those fields. But it is doubtful if filter-presses will ever solve the problem of the cheap treatment of low-grade slimes. It is generally allowed that the large filter-press, as at present constructed, is an unsatisfactory machine; its first cost is heavy, especially if transported any distance (a press holding a 7-ton charge weighs approximately 20 tons); it is a clumsy machine to handle, and an expensive one to operate. Yet, imperfect as it is, there is nothing just now to replace it. Managers perplexed with low-grade slime problems can do little more than experiment in the hope of finding something better, or at least wait until somebody else does. It seems not altogether improbable that with so many investigators at work on the problem, we may soon hear of a slimes process which will possess the great desiderata—high efficiency, comparative cheapness of installation, and low cost of operation.

The slimes at present overflowing from the cyanide tanks at the Liberty Bell mill are principally of clay, and, as already stated, extremely fine. With a system of pointed boxes and large 40-ft. settler vats, the experiments indicated that the slimes could not be settled in 48 hours to a sludge containing less than 80 per cent water. An experimental filter-press plant was installed, and operated in the following manner: A settler vat 5.5 ft. deep and 8 ft. in diameter was filled by a stream cut out of the total mill flow, equivalent to 2.5 gallons per minute. Into this stream milk of lime was fed in the proportion of 20 lb. lime to one ton of slimes. The vat required about 12 hours to fill, and then overflowed clear water for another 12 hours. At the 24th hour the slimes commenced to overflow, and the stream was shut off. The vat charge was then allowed to stand for 24 hours, the resultant pulp containing 15 per cent. of solids. This pulp was then agitated and fed into a small monteju, whence it was forced by compressed air into a 24-in. center-feed, recess-plate press of 19 plates. It took about five montejus to make a charge in the press; the filling was therefore intermittent, and the time required for the various emptyings of the montejus was 5, 15, 30,

50 and 80 minutes, respectively, or a total of about 3 hours. The pressure started at about 20 lb. and reached 80 lb. at the third monteju charge, continuing at 80 lb. to the end. The resultant cake on each side of the plate was only  $\frac{5}{8}$  in. thick. The water in the cake was then displaced by cyanide solution (0.3 per cent.), which took from 1 to 1.5 hours. The displacement was slow and imperfect; and the effluent liquor, until it reached standard strength, constituted a weak solution which was subsequently used for displacement in the second pressing. The effluent solution was tested constantly by titration as the solution was forced in, and this, together with the results of sampling the discharged cake, determined the displacement. The discharged cake contained from 30 to 40 per cent. of moisture; it was fed into a muller with strong solution and flowed thence to an extractor vat 5 ft. in diameter and 6 ft. deep, where about 1 lb. lime per ton of slimes was added to maintain the alkalinity. This pulp, containing 20 per cent. of dry solids, was agitated with air at intervals of 0.5 hour for 72 hours, beginning when the pulp first flowed into the vat, and ending when the last was drawn off. It was then forced through the press again at about the same rate and pressure as before. The strong solution remaining in the cake was displaced by the weak solution above referred to, and about 1.2 lb. of cyanide per ton of dry slime was left in the cake when discharged. About 0.5 lb. cyanide was lost in consumption during contact, making the total lost and consumed 1.7 lb. All solutions were run through zinc-boxes, and the zinc was finally reduced as a check. In this small plant a total of 7.7 tons (dry) was treated of an average value of \$3.84 (silver estimated at \$0.40 per ounce). The extraction by bullion yield was 74.5 per cent. As these tests were made under rather adverse conditions, it would be reasonable to expect a somewhat higher extraction in practice; and in the following calculations, 75 per cent. will be assumed as the probable working extraction from the slimes.

The mode of operating a large plant was to be as follows: The slimes product, after addition of lime, was to be divided between 20 pointed boxes, 5 ft. deep, 5 ft. wide and 20 ft. long, with constant underflow and overflow. Experiments indicated that this would eliminate about 50 per cent. of the water. Pos-

sibly a second series of boxes might be added to settle the pulp further. The final underflow was to be led to settler vats 40 ft. in diameter and 20 ft. deep, of which there were to be three in number, so that each vat-full of slimes would have three days between the beginning of filling and the end of emptying. In these vats, by settling, decantation, and possibly by the aid of filtering, it was thought that the moisture in the final pulp might be reduced to 75 per cent. All of the clear water was to be pumped back and re-used in the mill. These settler vats were to be provided with agitating apparatus to prevent settling and irregular supply to the presses during the process of press filling. From the bottoms of these vats, several 8-in. discharge pipes were to lead to the pumps; the latter to be a specially designed modification of the Gould triplex-plunger pattern, with ball valves, and provided with a water-flushing device to prevent erosion of the plungers. The pulp was to be pumped through a 4-in. pipe to a pressure tank on a hill behind the mill, the tank to be 10 ft. in diameter and 20 ft. high; the distance pumped to be about 400 ft., and the elevation about 180 ft., insuring 80 lb. pressure in the presses. The presses were to be fed directly from this tank. In practice, it was proposed to pump directly into the press and pipe column; and the tank on the hill was to serve merely to maintain a constant pressure,—to re-inforce the pump during the beginning of press filling, and later to act as a safety valve in case the pressure should exceed the limit. Some adjustment of this kind would be necessary, because the pumps were to be driven by belt and motor at constant speed, while the press would take a variable amount per minute, according to the stage of filling. These pressure tanks were to be provided with agitators if found necessary.

The battery of filter-presses was to consist of seven primary and seven secondary presses of the center-feed, recess-plate pattern, each press to have fifty 42-in. plates. This large battery was made necessary by the thinness of the cake, and by the slow operations of filling and displacing. After the formation of the cake, the residual moisture was to be displaced as in the test plant. The discharged cakes from the primary presses were to be dumped on a belt-conveyor running under a series of presses, and thence carried to mullers, where they

would be broken up and mixed with strong solution. The pulp from the mullers was to be elevated to the extractor vat, 35 ft. in diameter and 15 ft. deep, where treatment would be carried on with 80 parts solution and 20 parts slimes (dry). There were to be three of these vats, so that a 72-hour contact would be possible. The pulp was to be agitated at intervals in the extractor vats by stirrers or rotary pumps, and then pumped to the pressure tank on the hill (for which a pump would be used similar to that for the settler vats) and would be delivered thence to the battery of secondary presses. The effluent solutions were to be precipitated by means of zinc-dust. The discharged cake would contain about 30 per cent. of weak solution, which would be the principal cyanide loss.

The estimated cost of treatment in such a plant, treating 100 tons per day, is given in Table I.:

TABLE I.—*Estimated Cost of Treatment Per Ton on the Basis of 100 Tons Per Day.*

Items.		Cost Per Ton.
<i>Power</i> —1 pump, 150 gal., 200 ft. head,	12 H.P.	
1 pump, 150 gal., 200 ft. head,	12 H.P.	
Mullers,	25 H.P.	
Elevator to extractor vats,	5 H.P.	
Solution pumps,	10 H.P.	
Agitating settlers, etc.,	20 H.P.	
Miscellaneous,	16 H.P.	
Total,	100 H.P.	
100 H.P. at \$7 per month = \$700 per month,		\$0.23
<i>Cyanide</i> —1.75 lb. per ton,		0.43
<i>Lime</i> —20 lb. per ton,		0.06
<i>Zinc</i> —0.25 lb. per ton,		0.03
<i>Labor</i> —12 men on presses at \$3.50 per day,	\$42	
2 solution and pump men,	7	
2 shift bosses,	9	
1 roustabout, 10 hr.,	3	
Total,	\$ 61 per day,	0.61
<i>Canvas</i> —200 yd. per month,	100 per month,	0.03
Miscellaneous supplies and repairs,	300 per month,	0.10
Lights,	60 per month,	0.02
Heating,	30 per month,	0.01
Depreciation, 5 per cent.,	335 per month,	0.12
Insurance,	135 per month,	0.04
Miscellaneous and contingencies,	300 per month,	0.10
Total,		\$1.78

The cost of the plant, allowing \$5,000 for contingencies, was conservatively estimated at \$100,072, of which \$48,100 was the cost (delivered) of the presses alone.

The weak points in the method described above are no doubt obvious. The experimental work showed that it would be a slow process, requiring seven or eight hours for the cycle of operations in a press. The cakes formed, though only  $\frac{1}{8}$  in. thick, were almost impermeable; they could not be sufficiently dried, and it was found impossible to wash them thoroughly. These results led to additional tests on filter-pressing a pulp consisting of slimes, mixed with fine sand in various proportions. It was found, however, that the conditions were not materially improved until a mixture was used of about 100 parts of slimes to 250 parts of fine sand, the latter passed through a 200-mesh screen. This result naturally led to a consideration of the treatment in filter-presses of the whole product leaving the Wilfley tables, after re-grinding to a slime everything coarser than 200-mesh. This plan had much to recommend it. The extraction from the sands in the present leaching plant had been unsatisfactory; and as so large a proportion of the crushed product would already pass a 200-mesh (about 55 per cent.), the problem of re-grinding the remainder did not seem an overwhelming one. It was only necessary to determine whether the considerably increased extraction would offset the additional cost of filter-pressing, as well as the increased initial cost of a plant.

This new feature involved the re-grinding of about 120 tons per day. Various forms of grinding machines were considered, and the tube-mill finally chosen as best adapted to the purpose. It was ascertained that two of these mills, 5 ft. by 22 ft., requiring about 45 H.P. each, would do the work. A series of determinations made with a 6-in. laboratory press, which were confirmed with a 24-in. press, demonstrated that the maximum time taken up by the cycle of operations in a press would be about five hours, or that each primary and secondary press would have to be emptied five times in 24 hours. That this mixed pulp did not admit of more rapid pressing merely proved the impermeable and refractory character of the slimes. But there were several points in favor of the new method: A cake could be conveniently formed as thick as 2 in., admitting of the

use of the distance frame-type of press, of much larger capacity than a recess-plate press of corresponding cost; and the cake being more granular admitted of perfect displacement and thorough washing and drying to as low as 22 per cent. moisture.

For the double filter-press treatment of 266 tons per day, there would be required sixteen 42-in. 60-frame presses, holding 7 tons each, or fourteen 48-in. 50-frame presses, holding 7.5 tons each, or twelve 48-in. 60-frame presses, holding 9 tons each. The probability that the 48-in. press might be impracticable, left us to the selection of the 42-in. 60-frame press as the best size.

A series of tests on this mixed product indicated an extraction of from 70 to 85 per cent. For purposes of safe calculation, however, the probable extraction in practice was placed at 65 per cent.

The proposed method of treatment was, with few variations, practically the same as that for the pure slimes. The tailings leaving the Wilfley tables were to be run into spitzkasten, whose underflow would be led to the tube-mills for re-grinding. The material finally reaching the settling-vats would therefore be a combination of two products,—the re-ground sand from the tube-mills and the overflow from the spitzkasten. It was proposed to use the present leaching plant for purposes of settling. The five upper vats are 8 ft. by 10 in. deep; the lower vats, 9 ft. by 8 in. deep. It was found that by settling the slimes pulp in the upper tanks, decanting off the water, discharging the pulp to the lower vats and decanting off more water, a final pulp could be obtained in 48 hours which would contain only about 67 per cent. water. This material was then to be pumped to the high pressure tank and delivered to the presses as in the case of the pure slimes. Owing to the longer treatment required by the fine sands, four treatment-vats would be used instead of three, allowing a four days' contact with a 0.3-per-cent. solution.

The estimated cost of filter-pressing all the material from the Wilfley tables (266 tons), as above indicated, is given in Table II. on page 723.

The cost of erecting a re-grinding and filter-press plant, allowing \$5,000 for contingencies, was estimated at \$109,602, of which \$59,200 was the cost (delivered) of presses alone.

TABLE II.—*Estimated Cost of Treating all Product from Wilfley Tables (266 Tons).*

Items.	Cost Per Ton.
<i>Power</i> —Same as for slimes-plant, with addition of 90 H.P. for tube-mills, and 10 H.P. extra for mullers, or 200 H.P. at \$7 per month = \$1,400 per month, . . .	\$0.17
<i>Cyanide</i> —0.75 lb. per ton, . . . . .	0.1875
<i>Lime</i> —15 lb. per ton, . . . . .	0.09
<i>Zinc</i> —0.25 lb. per ton, . . . . .	0.03
<i>Labor</i> —20 men on pressers at \$3.50, . . . \$70	
2 solution and pump men, . . . . .	7
2 shift bosses, . . . . .	9
2 roustabouts, . . . . .	6
2 men for settlers and classifiers, . . . . .	7
1 helper, . . . . .	3
Total, . . . . .	\$102 per day, 0.38
<i>Canvas</i> —250 yd. per month, . . . . .	150 per month, 0.02
Miscellaneous supplies and repairs, . . . . .	300 per month, 0.035
Lights, . . . . .	120 per month, 0.015
Heating, . . . . .	60 per month, 0.0075
Insurance, . . . . .	200 per month, 0.025
Depreciation 5 per cent on \$140,000, . . . . .	583 per month, 0.07
Miscellaneous and contingencies (including labor on tube-mills), . . . . .	750 per month, 0.10
Total, . . . . .	\$1.13

Before passing to an estimation of comparative profits, we might indicate the three methods as follows:

1. The present practice of leaching the sands, by which 62.5 per cent. of the whole product is handled at a cost of \$0.65 per ton, the remaining 3,000 tons per month being run to waste as slimes; extraction of gold and silver from sands 50 per cent.

2. The proposed filter-press treatment of the slimes in conjunction with the above leaching plant, at a cost of \$1.78 per ton of slimes; probable extraction of gold and silver from slimes 75 per cent.

3. The abolition of the present leaching plant, the re-grinding of all sand coarser than 200-mesh size, and the treatment in presses of the whole product from the mill (about 266 tons per day) at a cost of \$1.13 per ton; probable extraction of gold and silver 65 per cent.

A study of the comparative values per ton, of ore, slimes and sand, from the manager's annual reports of the Liberty Bell Co.,



justifies the assumption that the value of the slimes per ton, produced by a given ore, will be about 50 per cent. of the value per ton of that ore; and that the sands maintain a proportionate value per ton of about 34 per cent. of the value per ton of the ore. This will serve to explain Table III., which gives the estimated net profits accruing from the different grades of ore.

TABLE III.—*Estimated Net Profits to be Derived from the Treatment of Various Grades of Ore by the Three Methods.*

Value of Ore.	Value of Wilfley Tailings.	Value of Slimes as Now Separated in Cyanide-Plant.	Value of Sands as Now Settled in Cyanide-Plant.	Profits Per Month in Cyanide-Plant.	Estimated Profits Per Month from Proposed Slimes-Plant.	Profits Per Month from Combined Sand- and Slimes-Plants.	Estimated Profits Per Month from Proposed Re-Grinding and Filter-Pressing.	Profits Now Obtained from Canvas-Plant.	Increased Profits Per Month by Re-Grinding Method Over Present Treatment.
\$10.00	\$4.00	\$5.00	\$3.40	\$5,250	\$5,910	\$11,160	\$11,760	\$1,000	\$5,510
9.50	3.80	4.75	3.23	4,800	5,340	10,140	10,720	925	4,995
9.00	3.60	4.50	3.06	4,400	4,770	9,170	9,680	850	4,480
8.50	3.40	4.25	2.89	3,950	4,220	8,150	8,640	775	3,965
8.00	3.20	4.00	2.72	3,550	3,630	7,180	7,600	700	3,450
7.50	3.00	3.75	2.55	3,100	3,060	6,160	6,560	625	2,935
7.00	2.80	3.50	2.38	2,700	2,490	5,190	5,520	550	2,420
6.50	2.60	3.25	2.21	2,500	1,920	4,420	4,480	475	1,905

It was important to determine how low-grade an ore it would be profitable to treat by adopting the method of re-grinding and filter-pressing all the mill tailings.

The total cost of treatment of the ore during 1902 was \$5.51 per ton.

Total cost per ton,	\$5.51
Cost of cyaniding per ton ore milled,	\$0.38
Cost of canvas-concentration,	0.18
Casualties,	0.42
	<hr/>
Cost to mine, tram, crush and concentrate,	4.53
Estimated cost of re-grinding and filter-pressing tailings,	1.13
	<hr/>
Total cost of proposed treatment,	\$5.66

Now considering that the total cost of treatment from October, 1902, to March, 1903 (inclusive), was only \$5.25 per ton, it may safely be assumed that with certain projected improvements and retrenchments, the cost might be brought as low as \$5 per ton. On this basis, there remains, after deducting the \$0.98

as detailed above, a cost of \$4.02 for mining, tramming, crushing and concentrating. Add to this \$1.13 per ton for re-grinding and filter-pressing, and a total cost of \$5.15 per ton is obtained. Then assuming 60 per cent. extraction in the present stamp-mill and 65 per cent. extraction from the tailings by the proposed new method of filter-pressing, an estimated total extraction of 86 per cent. from the ore will be obtained.

On the assumption of 86 per cent. extraction and \$5.15 cost per ton, a profit from various grades of ore would result as follows:—

Ore. Value Per Ton.	Profit Per Ton.	Ore. Value Per Ton.	Profit Per Ton.
\$10.00	\$3.45	\$7.50	\$1.30
9.50	3.02	7.00	0.87
9.00	2.59	6.50	0.44
8.50	2.16	6.00	0.01
8.00	1.73		

The foregoing statements show that a plant for sliming and filter-pressing all the tailings from the 80-stamp mill would be slightly more profitable than a slimes-plant added to the present leaching and canvas-plants; but that the initial cost would be greater. This excess of initial cost, however, would be more than offset by the profits, since in all probability the extraction by the sliming and filter-press combination method could be raised in practice above 65 per cent.

A conservative estimate places the profits from a combination-plant between \$20,940 and \$69,000 per year (on ore ranging in value between \$6.50 and \$10 per ton) over and above the profits obtained by the present system of leaching; this on an outlay of about \$110,000.

In offering the foregoing rather anomalous contribution, I am aware that I am presenting hardly more than the outlines of what, in the case in point at least, may be considered an unsolved problem. A definite record of results accomplished, such as may be expected eventually from filter-press workers in Australia, would no doubt be far more acceptable to metallurgists; and my sole excuse for publishing these observations is that they may be considered in the light of a query, and thereby provoke discussion of an important subject. The aspiration of more than one investigator has been to perfect a

cheap method of cyaniding slimes—an achievement which will have the obvious result of doing away with the present system of percolation, and revolutionizing milling methods by proving the expediency of fine crushing.

The experimental work at the Liberty Bell mine was conducted by Mr. Daniel Bosqui, the former cyanide manager, to whom I am indebted for much of the data presented, as well as for valuable help in the preparation of these notes.

My acknowledgments are also due to Mr. Arthur Winslow, General Manager, and Mr. Charles A. Chase, Superintendent of the Liberty Bell Gold Mining Co., for their generous co-operation.

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### Biographical Notice of J. Peter Lesley.

BY BENJAMIN SMITH LYMAN, PHILADELPHIA, PA.

(New York Meeting, October, 1903.)

#### *Abstract.*

[SECRETARY'S NOTE.—For lack of room this abstract, giving only an outline of Prof. Lesley's scientific work, is published here, instead of the full text of Mr. Lyman's account of his life, labors and character, already sent to members and associates in a separate pamphlet. Mr. Lyman's sympathetic and discriminating analysis, accompanied with abundant illustrative details of Prof. Lesley's remarkably versatile and fruitful career, completes the picture imperfectly sketched in the present abstract. Copies of the pamphlet may be obtained at the office of the Institute.]

THE subject of this sketch was born in Philadelphia, September 17, 1819, and graduated in 1838 from the University of Pennsylvania. His purpose of pursuing theological studies being hindered by the condition of his health, he took a subordinate position in the geological survey of Pennsylvania under Henry D. Rogers. His work there began in the sphere of topographical and geological map-making in which he was afterwards so eminent; and a map which he made of one field in 1840, before he was 21 years old, received high praise from Prof. Rogers in his official report of that year.

In 1841, resuming his earlier purpose, he entered the theological seminary at Princeton, N. J., and in 1844 he was licensed as a Presbyterian minister. But meanwhile he had done much work, outside of his theological studies, in compiling, collating and mapping for Prof. Rogers the rough field-notes of the Pennsylvania Survey; and after leaving Princeton he sailed for Liverpool, and made a pedestrian tour through England and parts of France and Germany, noting and recording many geological phenomena not previously recognized.

Returning, in 1845, he spent many months in the remoter valleys of Pennsylvania as a colporteur of the American Tract Society. But the fatigue and exposure of this occupation compelled him to abandon it; and in December, 1846, he resumed his work with Prof. Rogers, then at Boston, engaged in finishing the reports and maps of the Pennsylvania Survey.

In 1848 he returned once more to the ministry, and became pastor of a church near Boston; but, even while thus engaged, he continued to do occasional geological office-work for Prof. Rogers;<sup>1</sup> and in 1851 he resigned the pastoral office and finally devoted himself to the more healthful and less anxious practice of geology, assuming in that year, under Prof. Rogers, the charge of the office-work in the western-middle and southern anthracite fields, for the prosecution of which the Pennsylvania Legislature, after long indifference and delay, had at last made provision. He made a map between 20 and 30 ft. long, showing the surface topography, geology and underground mine-workings of the Pottsville region, from Donaldson and Tremont to New Philadelphia. This was never published; but Rogers' final report, printed in 1858, contained a compiled and reduced map of the whole anthracite region.

Meanwhile, Lesley made, in 1853-4, for the Pennsylvania R.R. Co., a map, 8 by 4 ft. in size, of the country from Johnstown to Greensburg, in western Pennsylvania, covering parts of Westmoreland, Fayette and Indiana counties. This work, a marvel of patient and skilful labor, exhibited the first use of contour-lines in geological maps. Even in Pennsylvania, where topography was so important as an index to geology, it had been previously indicated by hachures only.

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<sup>1</sup> See the paper "On the Importance of Surveying in Geology," *Trans.*, i., 183, in which Lesley's pioneer work in this department is stated, particularly on pp. 190 *et seq.*

Having become by such works well known as a topographer and geologist, Lesley was employed upon private surveys and reconnaissances of tracts in the Appalachian regions of Pennsylvania, Virginia and Tennessee. In 1855 and 1856 he made a minute survey and contour-line map of the Broad Top coal-field. In the winter of 1855-6, besides mapping the field-work of the preceding summer, he wrote the epoch-making book, *A Manual of Coal and its Topography*, a modest volume of 224 pages, which was published in August, 1856, and confirmed his reputation as a geologist of the first rank.

In the autumn of 1856 he became Secretary of the American Iron Association of Philadelphia, in which capacity, besides compiling and editing bulletins of statistical and other information concerning the iron-works of the United States, he prepared for the Association, and issued in 1859, *The Iron Manufacturers' Guide*, an octavo of nearly 800 pages, one-third of which gives brief notices of American works, while more than one-half discusses the geological occurrence of the iron-ores. Though, in part, a compilation, this book contained much original material, and is still a valuable authority.

In 1858 Lesley was elected Librarian, and in 1859 one of the Secretaries, of the American Philosophical Society. He remained Librarian until his resignation in 1884, and Secretary until he was elected Vice-President in 1887. The latter position he resigned in 1897. As Librarian and Secretary, from 1858 to 1884, he edited the Proceedings and Transactions of the Society, besides contributing to them about 65 formal papers (besides numberless impromptu remarks); and he devised and ultimately (in 1863, 1866 and 1878) published a classified catalogue of its library. During this period, moreover (in 1859), he accepted the chair of Mining in the University of Pennsylvania.

In the same year (1859) he superintended an extensive topographical and geological survey of part of the Coxe Estate, near Beaver Meadow, in the Pennsylvania anthracite region. His assistants in this work included Eckley B. Coxe, then a recent college graduate.

The *Second Geological Report of Arkansas* (1860) acknowledges the services of Lesley, secured "to superintend the press and revise the proof." This seems to have been but an "unconsidered trifle" of his manifold activity.

In 1861 he made for the Grand Trunk Railway a map, 15 ft. square, illustrating the iron-ore resources of the United States.

From 1860 to 1866 his services were in great demand, as the war of the Rebellion stimulated the industry of coal and iron. In 1862 he studied and described the principal coal-field of Cape Breton, N. S.<sup>2</sup> In 1863, at the request of the Pennsylvania Railroad Co., he crossed the Atlantic to study foreign methods of hardening the surface of rails, and the success of the Bessemer process. During this rapid trip through England, Belgium, France, Germany and Austria, he revisited the Jura, and, soon after his return, he definitely renounced<sup>3</sup> his former adherence to Rogers' opinion that the Appalachian erosion was effected by a cataclysmic deluge from the Arctic Ocean.

Lesley was perhaps the first professional geologist who investigated (in 1862, possibly even as early as 1859) the mode of the occurrence of petroleum in Pennsylvania,<sup>4</sup> and certainly the first to propound rational views on the subject. In 1863 he described in a paper before the Philosophical Society the new mineral Grahamite (a solid bitumen found in a fissure-vein in West Virginia); and in 1865 he examined and reported upon the oil-region of Gaspé, on the lower St. Lawrence.

Of the National Academy of Sciences, incorporated by Congress in 1863, he was one of the original members, and remained a member throughout his life.

In 1864 and 1865, the great demand for private and commercial geological surveys gave him much profitable but exhausting employment; but, in spite of threatening symptoms and occasional illness, he continued working through 1865. In 1866, however, complete nervous prostration compelled him to stop; and, with his wife, he spent about two years abroad. This period comprised a sojourn at Pau; three months at Paris (where his feeble health obliged him to abandon his work as a U. S. Commissioner to the Exposition of 1867); a pedestrian tour through the Vosges; a prolonged rest at Vevay, in Switz-

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<sup>2</sup> *Proceedings of the American Philosophical Society.*

<sup>3</sup> *Five Types of Topography, Transactions of the American Philosophical Society, xiii., 310 (1866).*

<sup>4</sup> See his article on "Coal Oil," in the *Report of the United States Department of Agriculture for 1862, pp. 429-447.*

erland; a journey up the Nile<sup>5</sup> and a return *via* Italy and England. He reached home in the spring of 1868, but was unable to do a full day's work for some years longer.

Nevertheless, in May, 1869, he added to his duties for the Philosophical Society and his occasional work on private geological surveys the editorship of the *United States Rail Road and Mining Register*, for which he wrote weekly acute and picturesque comments upon all sorts of topics, scientific, political, commercial, theological, etc., until the end of 1873.

In 1872 he became professor of geology and mining and Dean of the Scientific Department of the University of Pennsylvania; and in 1875 Dean of the new Towne Scientific School of that University.

He had for many years taken great interest in relief-map making. About 1854 he had made for the Pennsylvania Railroad Company a relief-map, with unexaggerated vertical scale, of the region to be traversed by the railroad in crossing the Alleghany Mountain, correcting the extravagant impressions produced by the enormously exaggerated profiles commonly used by civil engineers. Early in 1866 he made a model of the underground workings of the Hutchinson anthracite mine, at Plymouth, Pa. In 1873, at the University, with the help of Mr. John H. Harden, aided by some of the students, he had a number of relief-maps constructed; and, again, after becoming State Geologist, a number of others.<sup>6</sup> His long experience and sound geological judgment led him to protest strongly against the needlessness and harmful misleading of an exaggerated vertical scale in relief-maps. He justly said:<sup>7</sup>

"No relief of the surface is too delicate to escape the human eye when represented with sufficient skill and care in modeling. . . . If a relief-map be not true to nature, what is the good of it? Geologists have been forced to abandon exaggerated cross-sections; why should they permit relief-map makers to revive the discarded error, and put the representation of the whole in antagonism to the representation of the parts?"

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<sup>5</sup> On this trip, which extended to the first cataract, he and his wife were guests of the American Consul-General, Charles Hale, whose two sisters were also of the party. One of the latter was the author of "The Peterkins," in which delightful book Mrs. Lesley figures as the sensible "lady from Philadelphia."

<sup>6</sup> "The Construction of Maps in Relief," by John H. Harden and Edward B. Harden, *Trans.*, 1887, xvi., p. 298.

<sup>7</sup> *Science*, vii., p. 58, and *Trans.*, xvi., p. 295.

In 1874 he had a number of copies of half a dozen of his reports and papers on iron and coal-fields of the last preceding three or four years bound up together for private distribution, namely: Lyon, Shorb & Co.'s nearly 80 sq. miles of brown hematite lands in the Nittany Valley; South Mountain iron-ore lands; North Carolina Tuscarora iron-ore belt; the Embreeville (E. Tenn.) upthrow fault; the geological structure of Tazewell, Russell and Wise Cos., Va.; coal and iron lands at Ursina, Somerset county, Pa. The reports and papers are all handsomely illustrated with his characteristic topographical maps, sections and landscape sketches. This readiness of apt and pleasing illustration added greatly to the value and attractiveness of his reports. The volume is but a small collection of samples of a far larger number of pieces of like work that had been printed privately, or published as papers in the Philosophical Society. But much of the results of his personal field-work has never been printed, particularly the more extensive surveys: for example, his great contour-line map of the Kiskiminetas and Loyalhanna country for the Pennsylvania Railroad in 1853 and 1854; his Broad Top survey of 1855 and 1856; his Cape Breton coal-field surveys of 1862 and 1863; his surveys of the Tennessee coal-fields west of Knoxville; and others.

In 1874 he became the State Geologist of Pennsylvania, and began the Second Geological Survey of the State. He was now fifty-four years old, and besides his unusual natural qualifications and his thirty-five years of remarkable geological experience and reputation, his long and honorable connection with the first survey, his having constructed its State map from the mass of hitherto undigested materials, and his extraordinary familiarity from direct personal observation with the geology of all parts of the State clearly pointed him out to every geologist as the man of all men for the position; and the State was so fortunate as to discover the fact and secure his consent. Indeed, although the pecuniary compensation was small, compared with what his private practice would bring, and the responsibility was heavy and unceasing, he entered upon the new undertaking with enthusiasm. For he had through many years been anxious that a thorough survey should be made of his well-beloved, magnificent State, so richly endowed with useful minerals and so abounding in geological problems of the



highest theoretical interest and enormous practical importance; and he must long have been fully conscious that there was nobody in the world who could compare with him in fitness for taking charge of the work.

The Legislature, however, imperfectly aware of even the direct commercial value of such special fitness for work of that kind, and perhaps inclining rather to the opinion that the utility of geological surveys is generally doubtful, and at best remotely indirect, made only quite moderate appropriations, for a State of such immense mineral wealth, and those, after a few years, only from one session to the next. The assistants were, accordingly, for the most part young beginners; and in many cases, when they had gained experience, they were drawn away to less precarious and more lucrative work. At every meeting of the Legislature, special efforts must be made to enlighten it and induce it to continue the Survey. Mr. Lesley is said to have gone once in person to the Legislature; but was never willing to go again, showing himself "just like a child as to the way such business was done." He, doubtless, well illustrated what he had himself indignantly said in the *Rail Road Register* (in 1869, October 16), when some scurrilous newspaper had declared that the Academy of Natural Sciences was log-rolling for a plot of ground for its new building:

"A more helplessly innocent organization, for political purposes, cannot be imagined. They can log-roll in the woods for coleopterous insects; it would be rare fun to see their awkwardness if they were sent to Harrisburg to lobby for a donation of public land for themselves or anybody else."

He took pains to avoid one mistake of the first Survey, and did not let all the materials accumulate until a sudden cessation of the appropriations should leave it impossible to publish them at all for many years, while they were becoming, in great part, out of date. The work of the assistants was put in shape for prompt publication. In some cases they had not been so fortunate as to have the benefit of the best literary training; and responsible, as he unquestionably was, for the published form, and in some degree for the substance, too, he did a great deal of conscientious editorial work, making the reports intelligible and methodical, and even practically rewriting some of them. The mapping and the illustrations, in like manner, passed under his

close scrutiny and revision. In addition, he wrote long elucidating prefaces to most of the volumes in his usual vigorous, lively, clear, straightforward, trenchant style. There were, in all, 120 volumes of reports and map-atlases, besides Mr. Ingham's handy little index-volume for them all.

Yet with all this attention to the details, he still found time for several volumes by himself. He began the series of reports by *Report A*, an *Historical Sketch of Geological Explorations in Pennsylvania*, written at the beginning of the Survey, and published in 1876. It was in his usual lucid, cheery style, and contained much hitherto unpublished information in regard to the first Survey, and many valuable pertinent discriminating comments. It had been his intention to continue in the same volume a sketch of the history of other American State Surveys; but the pressure of other matter to be published interrupted the volume at the end of page 200, and, with the ever-accumulating mass of material of the new Survey, the convenient time for its resumption never came. In 1885 he published his *Report X*, a beautiful collection of geological maps of all the counties in the State, on a uniform scale of 2 miles to the inch, with uniform coloring, arranged in octavo form, and preceded by text of his own, briefly pointing out the leading geological features of each county.

In 1889 and 1890 he issued *Report P 4*, a *Dictionary of the Fossils of Pennsylvania*, in three volumes, compiled by himself. It contains illustrations of all the known fossils of Pennsylvania, except the flora, already reported upon by Lesquereux and Fontaine; and all are arranged alphabetically, for the most convenient reference from the county reports, where they are mentioned by the different assistants. This is a handy reproduction of an immense number of fossil figures gathered together from many widely scattered sources which are carefully referred to; in many cases, sources that are out of print and difficultly accessible, and in some instances rare. Professed paleontologists at first looked askance upon a work in their own field by one who did not profess to be one of them, and a work not zoologically arranged; but in time they came to value it more highly, and Dr. Clarke, the State Paleontologist of New York, has, in his biological enthusiasm, claimed that the *Dictionary* is more sought for than any of the other reports of the

Pennsylvania Survey. This is, however, a decided (of course, unintentional) exaggeration, though not without some shadow of a foundation. Mr. Lesley had always had the highest admiration of paleontology, though aware that in Pennsylvania its immediate usefulness in helping to work out the geology was not generally so important as in some other regions. He bore in mind that<sup>8</sup>

“When the geological survey of Pennsylvania was first ordered, its first business was well understood to be not scientific, but practical. It was to study and to find out all about the iron, coal, oil, gas and other mineral resources of the State. . . . This task the Survey has faithfully and zealously performed for fifteen years; and its strictly practical character is acknowledged by those intelligent business men who are the wealth-producers of the State.” Nevertheless, he adds: “I had always realized that the Survey would leave unperformed one of its necessary tasks if it did not fully explain the fossil geology of the State, as a supplement to its mineral geology; but the practical work of the Survey was so heavy that any adequate report of its fossils had to be left to the very last.”

He himself also undertook the writing of the *Summary Final Report*, a condensation and methodical rearrangement of the results of all the previous reports; and he accomplished two volumes out of the four, and nearly half of the third (or of the first part of the so-called third, which is bound in two thick volumes). His health then (in 1893), under such long-continued heavy strain, with his increasing years, broke down so completely that he had to give up work almost entirely, and the report was finished by Messrs. E. V. D’Inwilliers and A. D. W. Smith, with only comparatively slight occasional consultation with him. It was naturally disappointing to Mr. Lesley to be unable to lay the cap-sheaf upon the greatest work of his life; but to his kind soul the bitterest regret was that he was unable, in doing so, specially to show his warm appreciation of the aid he had received from his assistants. One of them, however, Prof. J. J. Stevenson, in speaking of the *Report*, says:<sup>9</sup>

“In this marvelous compilation he gave a synopsis of every assistant’s work, according unreserved recognition to each observer, and frequently showing an unselfish neglect of credit due to himself for earlier discovery of facts and determination of principles.”

All the assistants, with scarcely an exception, gratefully appreciated his never-failing generous consideration towards every one of them; and some were ready to adore him.

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<sup>8</sup> *Report P 4*, p. vi.

<sup>9</sup> *Science*, vol. xviii. (new series), p. 3, July 3, 1903.

The 120 volumes of reports and atlases of the Survey are, indeed, his great "monument more lasting than brass." As he truly says, the practical character of the Survey was always kept in view, and the constant aim was to gather knowledge that would have a useful bearing upon the working of the coal, iron, oil, gas and other minerals of the State. Invaluable work was done in the detailed survey of the anthracite region, under the able direct management of Mr. Charles A. Ashburner at first, and later under Mr. Frank A. Hill; with an extremely careful compilation of all the maps of the underground workings of all the mines, and with much of the surface topography, all reduced to one scale and published on a great number of sheets, with numerous cross-sections and columnar sections. Such work is not of general popular interest, and is little seen outside of the coal regions; but is there of the utmost utility. The bituminous coal-fields, the oil- and gas-fields, and the iron-ores were also carefully treated. Of course, in such fields, and to some degree in all other parts of the State, mining and exploration are constantly going on, new facts are daily coming to light; and it was Mr. Lesley's earnest desire that there should be a permanent geological survey established for the perpetual gathering and recording of such useful information, which, if not properly gathered and recorded, soon becomes lost forever. He was as far as possible from imagining that his Survey was a finality, or that every important fact had been discovered, and every serious problem solved.

He not only aimed to have the Survey work and investigations of an entirely practical character, but to have the results reported in language that would be simple, clear and readily understood. He appears never to have ransacked the Greek and Latin dictionaries to construct new-fangled words for the expression of novel ideas or for the adornment of new classifications, nor did he undertake to reform our ordinary English language by transferring poetical words to prose; though with appreciation he acquiesced in devices of that kind deemed advisable by others. It could not have been ignorance that kept him from thinking to propose such changes. On the contrary, it is more likely his good knowledge of Greek and Latin and poetry and English made changes of that kind seem unnecessary or distasteful. At any rate, the older he grew, the more

he aimed at using the simplest language; and he sometimes took special pains to devise a good Saxon equivalent of some sesquipedalian term derived from the classical tongues. His good schooling, too, and ample experience seem to have led him to look with little sympathy upon various minor so-called scientific reforms, that appear important to the less thoroughly trained.

He found time during the eighties to write a number of outside pieces, notwithstanding his geological and ordinary Philosophical Society work. He had, owing to the heavy stress of his other duties, ceased teaching at the University in 1878, receiving in 1886 the honorary title of Professor Emeritus, for the rest of his life. He published a second edition of *Man's Origin and Destiny*, with six new chapters, in 1881. In 1882 he published *Paul Dreifuss, His Holiday Abroad*. He edited in 1885 the *Early Proceedings of the American Philosophical Society from the Original Records, 1744-1838*, a thick octavo volume. He contributed, in collaboration with Rev. Charles G. Ames, the article on "Pennsylvania" in the *Encyclopædia Britannica*, published in 1885; and wrote a number of articles for such periodicals as the *Forum* or *Nineteenth Century*, as well as for newspapers. But all these prolusions, that would have been serious work to some men, seemed to be for him mere trifles, safety-valve escapes of superfluous steam.

The book, *Paul Dreifuss*, gives a photographically minute account of a holiday trip in the winter of 1881 and 1882; one of several he made to Europe to prevent a complete breakdown of his health in consequence of his constantly overdoing his strength. The trips were for rest, literally recreation, and he generally spent each time only about two or three weeks on the other side; for instance, in 1874 (July 13-September 2), 1876, 1878, 1880, 1882, 1884, 1887 and 1888. The book was published as the work of John W. Allen, Jr.; but many years afterwards was attributed to Mr. Lesley in two or three biographical dictionaries, evidently on authority from him or his family. It not only describes minutely his own doings and philosophical and witty reflections during a short trip to Europe, but recalls many of his earlier experiences in Europe and elsewhere, particularly his European journey of 1863.

In 1884 he served as President of the American Associa-

tion for the Advancement of Science, at the great Philadelphia meeting; and in 1885 made the retiring presidential address at the Ann Arbor meeting. The address was devoted chiefly to setting forth, with good judgment and graceful wit, views of general importance, as a scientific man of long experience: on the noble, unselfish character of the true, but rare, philosopher, or superior man of science; on the dangers of too great accumulations of mere undigested, unorganized scientific facts; on the risk of smothering wisdom, the spirit of science, with its letter, inordinate learning; on the utility of science to the world; on the need to diffuse, not so much knowledge, as the skill to discover it; on the excessive frequency of forward tyros who imagine they have that skill; and on the manifold advantages of "dead work" in science, or the drudging, tedious collection and thorough digestion of the elementary facts of investigations, its absolute necessity to investigators, teachers and professional experts, the invigorating mental effect of its habitual performance, and its reposeful, refreshing effect on the tired brain. He spoke of this from ample experience; for he himself had patiently waded through such labor in vast quantity, with unflagging steadiness, in addition to his higher, more conspicuous work. He could, in fact, often have called himself, as Johnson in his dictionary defined the lexicographer, "a harmless drudge," and with still greater reason.

He interested himself greatly in the improvement of the apparatus and methods of surveying, chiefly to lessen its drudgery, so far as possible. About 1854<sup>10</sup> the Pennsylvania Railroad, under his guidance, made a very accurate railroad odometer, and measured the whole line from Philadelphia to Pittsburgh, fixing the mile-posts anew. He also used an improved odometer in surveying, and proved its superiority to chaining, both in convenience and exactness, even off the roads, on rough ground. This odometer was in use in 1855 and 1856. Some odometers, too, were made for use on his State Geological Survey. The aneroid was another instrument to the improvement of which he gave much attention. About 1857 he devised one that was made for him by the skilful Becker, and was in successful use for many years. It had twelve vacuum-boxes and a dial plate

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<sup>10</sup> *U. S. Rail Road and Mining Register*, September 4, 1869.

14 in. in circumference, corresponding to only 2 in. of the mercurial column, or about 2,000 ft. of height; and was well compensated and light. In 1862 he experimented a little with stadia measurement, and was convinced of its advantages; but encouraged a young assistant to elaborate the method further within the next two or three years. A highly convenient micrometric pair of dividers for plotting field-notes was an invention of Mr. Lesley's in 1868, and was described by him in a paper read before the Philosophical Society in April, 1873, and republished in the *Rail Road Register*, June 28, 1873. It was so published in order to save his fellow topographers from any risk of having to pay a royalty to anybody else, though he did not desire a patent upon it himself.

In 1886, busy as he was, he with great good humor consented to deliver the Commencement-day oration at Smith College, Northampton, Mass.; and it proved to be highly acceptable.

Mr. Lesley was not only an Honorary Member of the American Institute of Mining Engineers (1890), and corporate member of the National Academy of Science, and Fellow of the American Association for the Advancement of Science, as already intimated; but he was a member of the Academy of Natural Sciences of Philadelphia; of the Boston Natural History Society; the American Oriental Society; the Oriental Club of Philadelphia; Foreign Honorary Member of the American Academy of Arts and Sciences; Foreign Member, Geological Society of London; Associate Member, Société Géologique du Nord; Member of the Moscow Imperial Society of Naturalists; the Emden and Neufchâtel Academies of Science; and the Lille Academy of Natural Science. In 1878 the degree of LL.D. was given him by Trinity College, Dublin.

During the seventies, eighties and nineties, he was conscious that some special care should be taken of his health, but imagined that he was doing all that was needful in the way of exercise if he walked every evening half a mile to the Union League club-house and back, and played billiards there for an hour or two; which would clearly have been an insignificant amount of exercise for a man who had not a constitution naturally so unusually vigorous as his. It is not surprising, then, that with his excessive industry he should have quite broken down in

1893; on the contrary, it is astonishing that he should have held out so long. At length, in the summer of 1896, when the last volume of the reports and atlases had been published, and his health had not been recovered, and he could do no work whatever, it was decided to give up his home in Philadelphia and stay permanently at Milton, Mass., where he had for some years spent the summer, the village of his pastorate nearly fifty years before, and the home of some of his wife's relatives. His brain was affected much as it had been in 1866, but his increased age now made it impossible, as it proved, to restore it even with a few years' rest; and though serious work was wholly given up, he gradually failed still further. In other respects, however, his health was generally not such as to cause discomfort, yet with occasional ill-turns. He had every needful care and fond attention, and was in the main comfortable. He passed the time mostly in listening to reading aloud, which had been his favorite recreation for a portion of each day for many years. He himself was a delightful reader-aloud; but now he preferred listening. The reading was mostly light, the best novels; but as late as the autumn of 1901, he was listening to Curtius's *History of Greece*, and making excellent comments upon it. It might, however, be said that he was continually failing, in fact, slowly dying for eight years or more. Finally, on the morning of June 1, 1903, he had a sudden stroke of apoplexy, and he continued unconscious until his last breath, at about eleven o'clock in the evening of the same day. His funeral took place on June 4th, and, in accordance with his own desire, his remains were cremated. He was in his eighty-fourth year, and had outlived his sister and all his brothers; but his wife, two daughters (Mrs. Charles W. Ames, of St. Paul, Minn., and Mrs. Henry K. Bush-Brown, of Newburgh, N. Y.) and ten grandchildren survive him. With his wonderful power, vigor, activity, brilliancy and kindness, he had attained in extraordinary measure "the five blessings" of the Far East: long life, prosperity, honors, love of virtue, and a peaceful end. What is more than all that, the well-merited new life he has begun to live in these descendants surely equals, if not exceeds, any anticipations or hopes he could have entertained.



## Biographical Notice of John F. Blandy.

BY R. W. RAYMOND, NEW YORK CITY.

(New York Meeting, October, 1903.)

THE death of John Frederic Blandy, which occurred September 17, 1903, at Prescott, Arizona, terminated the earthly activity of one of the most active and able of the American mining engineers of the last generation.

Mr. Blandy was born, April 24, 1833, at Newark, Delaware. His father was an Englishman, who had been in business in the island of Madeira, and had come thence to settle in the United States.

At the age of eighteen, he was attached to the engineering field-corps of the Philadelphia & Reading Railroad; but, after a year of this service, he went to Germany, where he spent three years, entering in 1852 the Mining Academy at Freiberg, Saxony. Graduates of that institution need not be told that, at the period referred to, it held unique prominence in its special field, not only by reason of the great investigators and instructors, such as Cotta, Breithaupt, Weisbach, Plattner and Gaetschmann, who belonged to its faculty and brilliantly blended theory and practice in its work, but also because rival institutions were few. It cannot be fairly said that Freiberg has fallen below its ancient standard of excellence. Its chairs of instruction are still occupied by eminent men, worthy successors of the illustrious teachers who preceded them. But a host of technical academies, especially in this country, munificently equipped and efficiently conducted, have come into existence during the last half-century; and no one institution can ever hereafter claim the pre-eminence which the Freiberg Mining Academy once enjoyed. In those days, American mining engineers returning with satisfactory credentials from that school were sure of immediate employment.

Mr. Blandy justified at once the presumption of professional ability attached to a graduation at Freiberg. In 1855, upon his return to the United States, he was engaged to go to the Lake Superior copper-region, where he remained for eight years, representing, among other clients, the interests of the Paris house of Rothschild. His thorough knowledge of the topography, geology and exploitation of the Lake Superior copper-deposits was shown in several published papers (cited below), and others of which I have heard, but have not been able to obtain the precise titles and dates.

Returning in 1863 to Pennsylvania, he became connected with the Little Schuylkill Coal Company, of which he was for many years president and general manager, and which, through his efforts, was finally absorbed by the Reading Coal and Iron Company. In 1880 he went to Arizona, where the rest of his life was spent. In 1889 and 1890, he held for some time the position of Territorial Geologist; and he frequently made the compilation of mining statistics for the United States Geological Survey. As a contributor of many able articles to scientific and mining journals he was well known; and his opinions on mining property and mining operations were often asked and highly valued.

Mr. Blandy was one of the original members of the American Institute of Mining Engineers. Though prevented from being personally present at the first informal session, held May 16, 1871, at Wilkes-Barre, Pa., he had signified his desire to co-operate in the organization of the new society; and, on the following day, his name was reported with approval by a special committee, and he was duly elected. Moreover, in recognition of his professional eminence, he was elected, at the same meeting, a vice-president of the Institute. In this capacity he served for three years, and again, upon re-election in 1878, for two years—the full term under the new rules, adopted in 1875. His contributions to the earlier volumes of the *Transactions* of the Institute were valuable, both by virtue of their intrinsic merit and as an evidence of his ardent interest in the young society. The following is a list of them in chronological order:

Title.	Transactions.		Date.
	Vol.	Page.	
Topography, with Especial Reference to the Lake Superior Copper District, . . .	I.	75	1871
Stamp Mills of Lake Superior, . . .	II.	208	1874
Remarks on the same subject, . . .	II.	299	1874
On Evidence of Streams During the Deposition of the Coal, . . . . .	IV.	113	1875
The Use of Anthracite Waste, . . . .	V.	465	1877
The Lake Superior Copper Rocks in Pennsylvania, . . . . .	VII.	331	1879
The Mining Region around Prescott, Arizona, . . . . .	XI.	286	1883

Most of these contributions were the carefully elaborated results of long observation, and will remain permanently authoritative, apart from their immediate interest at the time of their publication. The list, as a whole, bears witness that, in every new sphere of his professional activity Mr. Blandy freely gave to his professional colleagues the results of his own experience. The later years of his life were full of suffering, which he endured bravely, submitting to repeated surgical operations, and finally, after having survived them all, dying from the effects of a fall which, in his feeble condition, proved fatal.

I knew him well for many years as an accomplished and thoroughly upright expert, a genial and generous companion, and a loyal friend.

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### The Electric Steel-Furnace at Gysinge, Sweden.

BY F. A. KJELLIN, GYSINGE, SWEDEN.

(New York Meeting, October, 1908.)

THE problem of smelting steel by electricity attracted years ago the attention of inventors, and as early as 1879 C. W. Siemens constructed his first furnace for the smelting of metals, especially steel. His furnace was of the arc type in which the voltaic arc was formed between a carbon electrode and the metallic content of a crucible, the length of the arc being regulated by an electromagnetic device.

This electric furnace, as well as others of similar construction, has the inconvenience that the source of the heat—the voltaic arc—possesses a temperature of 3,700° C., which is much

more than that required by steel-smelting. The result of the use of this intense heat is that the steel is overheated in the vicinity of the arc, while in other parts of the furnace it has not yet reached the temperature suitable for teeming. Another inconvenience is that the steel very easily takes up impurities from the electrodes consumed.

The cost of the electrodes also is an item that sensibly increases the cost of production, and the carbon monoxide resulting from the oxidation of the electrodes exerts a bad influence, as it prevents the steel from giving off the carbon monoxide dissolved in it. A more uniform heating of the steel can be obtained by passing electric currents of great intensity through the steel, and using the heat evolved by the resistance of the steel for the smelting. But as the resistance of metals, even when molten, is comparatively low, the currents used must be so great that copper cables get a section at least as great as that of the steel in the furnace.

Mr. de Laval, the well-known Swedish inventor, has tried to diminish this inconvenience by using molten slag instead of steel as resistance in his furnace; but the results cannot have been satisfactory, as the patent has been allowed to expire.

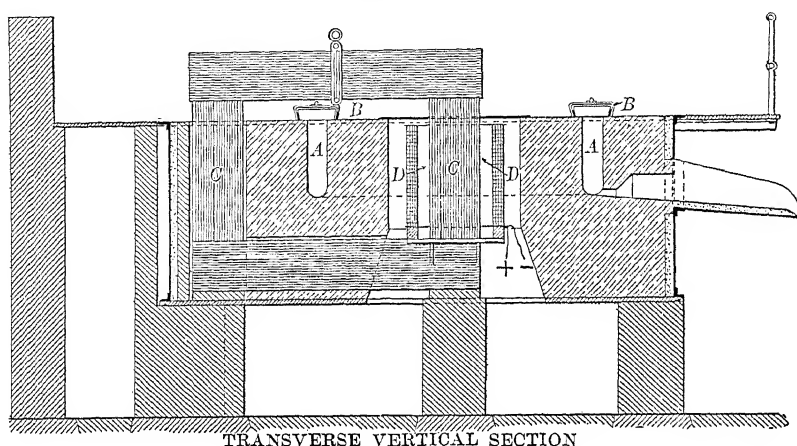
The greatest difficulty in electric furnaces of this kind comes from the electrodes, for the reason that carbon electrodes have high resistance, cause great losses of tension and power, and are soon consumed at the contact with molten steel. Water-cooled iron electrodes could possibly be used, but then the magnetic properties of the iron create new difficulties. In order to get sufficient intensity of current, alternating currents must be used, and then the magnetism causes the concentration of the current at the surface of the electrodes (skin-effect), and the result is great current-density in the electrodes with great losses of power, and by the influence of self-induction a diminished capacity of the electric generator to convert mechanical energy into electric energy.

In order to eliminate these difficulties I proposed to Mr. Benedicks, general manager of the Gysinge works, in May, 1899, to build an electric furnace at Gysinge, without electrodes. My project was accepted, and I went to Gysinge to carry it out.

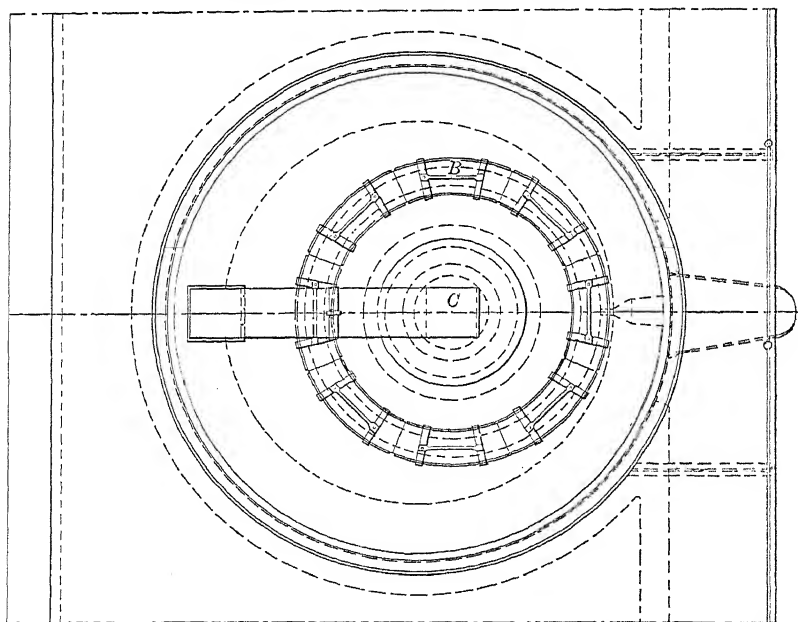
The illustration, Fig. 1, shows the principle of the furnace.

An annular groove *A A* forms the furnace room, the sides and bottom of which consist of refractory bricks. The covers *B B* close the furnace. In the center of the circle formed by the

FIG. 1.



TRANSVERSE VERTICAL SECTION



PLAN

PLAN AND TRANSVERSE SECTION OF THE KJELLIN ELECTRIC STEEL-FURNACE.

furnace room is a quadrangular core or cone, *C*, formed of thin insulated copper wire, which continues outside the furnace, and forms with the furnace room the two links of a chain. The

coil *DD* is connected with the poles of an alternating-current generator.

When passing through the coil the current excites a varying magnetic flux in the core or cone, and the intensity of current in the steel is then almost the same as the primary current multiplied by the turns of wire in the primary coil. The tension is naturally reduced in almost the same ratio as the intensity is increased. In this way it is possible to use an alternating-current generator of high tension, and yet obtain a current of low voltage and great intensity in the furnace, without using transformers with copper cables of large sections and powers, and costly electrodes.

In February, 1900, the first furnace at Gysinge was ready for use, and by March 18 the first steel ingot was cast. The ingots from the beginning were of an excellent quality. The problem was thus solved technically, but not economically, for, with an electric generator of 78 kw., only 270 kg. of steel were produced in 24 hours.

The next furnace built was ready in November, 1900, and produced, with 58 kw., from 600 to 700 kg. of steel-ingots in 24 hours. The charges weighed 100 kg., and the time between the teemings was from 3 to 4 hours. The output was not fully satisfactory, because the cooling-surface of the walls was too great compared with the contents of the furnace; also the cost of repairs was rather high.

On August 11, 1901, the sulphite pulp-mill of Gysinge was completely destroyed by fire, and it was then resolved not to rebuild it, but to use for steel-smelting the water-power formerly absorbed by the pulp-mill.

The new plant was ready to start in May, 1902, and has since that time worked satisfactorily. It consists of a furnace containing 1,800 kg. of steel from which from 1,000 to 1,100 kg. are taken out at each teeming, and the rest left to keep the current passing. The furnace produces with 165 kw., or 225 e.h.p., 4,100 kg. of steel-ingots in 24 hours when charged with cold materials. The electric generator gives an alternating single-phase current of 3,000 volts, which is directly transformed, by means of the primary coil and iron core (cone) of the furnace, into a current of about 30,000 amperes in the steel that forms the secondary circuit.

The smelting-process, as carried on at Gysinge, where only first-class steel is produced from the excellent Dannemora pig-iron and weld-iron, is as follows:

After teeming, about 800 kg. of metal is left in the furnace to keep the current passing, and to this are added such quantities of pig-iron, bar-ends and steel-scrap as experience has proved will give the percentage of carbon desired in the resultant steel. In teeming there is always less carbon in the steel than was contained in the materials charged, for the reason that the pig-iron is rusty and the steel also takes up a little silicon by reduction of the silicic acid of the lining. When the charge is molten and over-heated to a suitable degree, the teeming is done in the same way as in the open-hearth furnace, by making a hole in the wall. The upper part of the furnace, as shown in Fig. 1, is on the same level as the working-floor, and the charging is done simply by taking off the covers and putting in the materials. As the heat is produced in the steel itself, the slag is not so hot as in other steel-furnaces, and consequently the workmen do not suffer much from the heat. The steel produced is, as mentioned above, of an excellent quality, uncommonly dense, unusually homogeneous and tough, easy to work cold when annealed, and less disposed to crack and warp in hardening than other kinds of steel. I think that the cause of these excellent qualities, apart from the good raw materials used, must be due to the fact that the product contains less dissolved gaseous matter than other kinds of steel.

As every one knows, even small quantities of gases, especially hydrogen, injure the quality of steel; and the principal cause of the high quality obtained by smelting in crucibles is, in my opinion, that the steel is not in contact with the fuel gases, though it may absorb a little through the porous walls of the crucible. In the electric furnace described above, the steel has no opportunity to take up such gases or other impurities, and the quality is even better than that of crucible-steel with the same composition.

To make special steels with nickel, tungsten, or chromium, offers no difficulties, and the alloys themselves are quite homogeneous. The cost of production depends principally on the efficiency of the furnace and the price of the power. In the

furnace now in use at Gysinge, the losses have been proved experimentally to be 87.5 kw., so that the effective power absorbed by the steel is  $165 - 87.5 = 77.5$  kw.; and, as those produce 4,100 kg. of steel in 24 hours, one effective kilowatt produces about 53 kg. of steel-ingots in the same time. Every additional kilowatt in the furnace, when the size is not altered, increases the output by 53 kg., and it is calculated that within a few months, after a stronger water-wheel is obtained, 200 kilowatts will produce about 6,000 kg. of steel-ingots in 24 hours.

As the absolute cost of labor and repair will be the same, those costs for 1 ton of steel-ingots will be about two-thirds of the present cost, and the price of power per ton also will be sensibly diminished. At Gysinge the cost of repair (renewing the lining of the furnace when it is worn out) was \$60.

From experience with this furnace it is calculated that a furnace of 736 kw., or 1,000 e.h.p., will produce 30,000 kg. of steel-ingots in 24 hours, when charged with cold materials. With hot materials the output is much greater. For instance, if 250 kg. of molten pig-iron are charged for each ton of steel-ingots produced, the output is increased from 30,000 to 36,000 kg. in 24 hours, with 1,000 electric horse-power.

In my opinion, the costs of labor and repairs for a furnace of this type will be less than those of an open-hearth furnace of the same size, so that, where power is cheap, there is a possibility of producing a steel which can compete with the expensive crucible-steel, at a smelting-cost not exceeding that of steel made in the open-hearth furnace.

SECRETARY'S NOTE.—After the sheets containing the first part of this paper were printed, the following correction was received: On p. 744, line next to the last, "copper wire" should be "iron sheets".—R. W. R.



## The Power Plant of the Moctezuma Copper Company at Nacozari, Sonora, Mexico.

BY JOHN LANGTON, NEW YORK CITY.

(New York Meeting, October, 1908.)

### INTRODUCTION.

IN determining what kind of power should be adopted for the works of the Moctezuma Copper Co. at Nacozari, Sonora, Mexico, certain considerations imposed by local conditions had to be taken into account.

*Water.*—During most of the year the water-supply is abundant for all purposes, but it dwindles rapidly when the dry season sets in, reaching a minimum flow which is entirely underground. It was evident, that with the greatest care in keeping down the unavoidable loss of water in milling and smelting, these operations and hence the capacity of the works would ultimately be limited for about two months of the year by the small water-supply available at the period of minimum flow.

*Transportation.*—The nearest railway terminus was in Arizona, at the international boundary-line; and the price of supplies at that point would be enhanced at Nacozari by the additional cost of a 90-mile wagon-haul. During the rainy season, transport to Nacozari was much impeded, and at times made impossible, by the condition of the roads. Hence, to insure continuous operation at the works, it would be necessary to provide for hauling the year's supplies, including fuel for power and coke for smelting, in less than eleven months. In other words, the average freight delivered daily at the works should be at least 10 per cent. more than the average daily consumption.

For moderate quantities of freight, pack-carriage by burros is astonishingly cheap. But it has a capacity-limit which is generally very soon reached. A 200-lb. load, and 10 miles a day, 5 with and 5 without load, is the full capacity of a burro. This means one burro per ton-mile of the round trip; and,

at this rate, the number of animals to be handled, especially at the unloading-point, and required to feed themselves off the country, soon becomes prohibitive.

There is also a capacity-limit to wagon-transport, set by the number of teams available without undue interference, both on the road, and at loading- and unloading-points.

*Fuel.*—The country surrounding Nacozari, being sparsely wooded, mainly with scrub white oak, offered a source of fuel-supply of considerable extent, though of poor quality. But the character of the country is so mountainous and broken, that even where a few main wagon-roads were feasible without costly road-making, the cut wood could be collected only by burro pack-trains, and so brought to the wagons; while from a great portion of the wooded territory only pack-trails to Nacozari would be possible.

For equal heat-values, wood at Nacozari cost at that time much less than coal; but it was decided that a sufficient quantity of wood-fuel for steam-power was not available, and that a large proportion of coal would still be needed for boiler-fuel.

The total power contemplated to begin with was from 500 to 600 h.p., of which the concentrator, the converter blowing-engine and the furnace-blower would be the principal consumers. Compound condensing steam-engines of moderate power would be the highest grade of steam-apparatus appropriate to the size and location of the plant. And, including shrinkage, 3 lb. of coal per brake-h.p.-hour would be a very good result from such a plant. In comparison with this result, 1.5 lb. of coal per brake-h.p.-hour was obtainable with gas-engines using producer-gas. Moreover, besides the greatly reduced fuel-cost, gas-power offered very important additional advantages in facilitating the continuous operation of the works at their full capacity: by cutting down the water-consumption in the dry season; by reducing largely the stock of fuel needed to tide over periods of interrupted transport; and by reducing the average quantity of daily freight.

The effect in alleviating transport-difficulties would be increasingly beneficial with that extension of capacity which always follows successful operation.

The fuel-economy promised by gas-engines had led to their

use in supplying all the power needed for a small electrolytic refining-plant, operated by the Copper Queen Consolidated Mining Co., with the view of giving them a practical trial, which would test their mechanical suitability for the exacting conditions and continuous use incidental to the power-service of a copper-reduction works.

When it became necessary to decide the question of power for Nacozari, more than a year's experience of this plant had led to the conclusion that a good gas-engine in good condition is a simple and reliable prime-mover, requiring no better class of attendance than steam-engines of the same size; and to the further conclusion, that good condition is easily maintained by periodical inspection and cleaning, for which provision must be made.

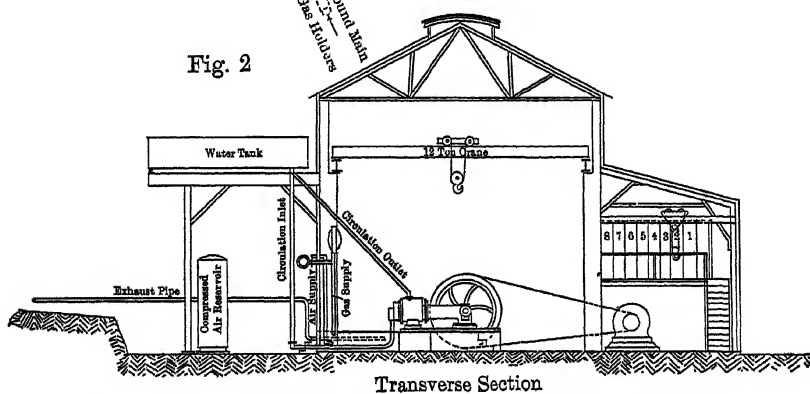
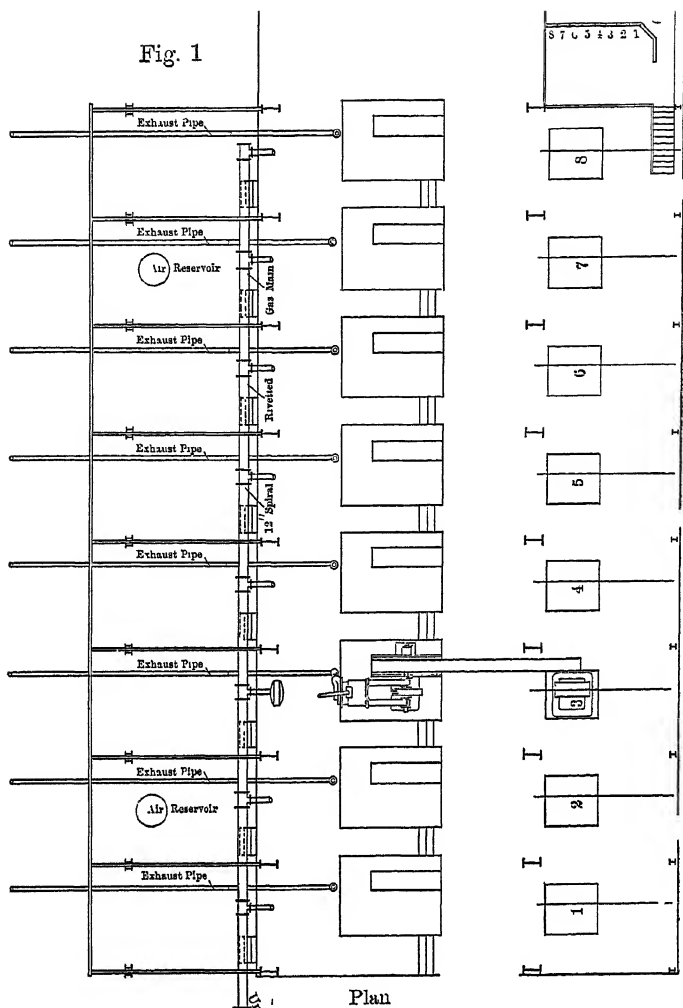
On consideration of the whole question, with due weight given to the facts briefly indicated above, it was decided that properly selected gas-engines, in a suitably arranged power plant, would be well adapted to the general conditions at Nacozari, and would give a fuel-economy which was there particularly valuable, in view of the limited local supply of wood and the then necessarily high price of coal.\*

#### DESCRIPTION OF POWER-HOUSE.

The power-house, the plan and elevation of which are shown in Figs. 1 and 2, and the interior view in Fig. 3, contains eight single-cylinder Crossley gas-engines, 18.5 in. in diameter by 24-in. stroke, running at 200 rev. per min. These engines are of the four-cycle type, with "hit and miss" governor. Each engine is direct belted to a 65-kw. direct-current generator. The generators are run in parallel, and deliver to the main switchboard in the power-house, from which the current is distributed at 250 volts to about 40 different motors scattered about

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\* After eighteen months' operation, the cost and difficulties of transport were materially reduced by the construction of the Nacozari Railroad to Cos, 22 miles from Nacozari. Its extension to Nacozari, which is now under contemplation, will further reduce the freights on coal. Meanwhile, the cost of gathering wood has increased with the exhaustion of the supply in the immediate vicinity; so that, when the railroad is completed to Nacozari, the two fuels will cost approximately the same for equal heat-values.



PLAN AND TRANSVERSE SECTION OF THE POWER-HOUSE AT NACOARI.

the works, ranging in size from 5 h.p. to 175 h.p. Arc and incandescent lights are operated from the same circuit as the motors.

The engines are started by compressed air, a method which has been found very simple and convenient. The cock, by which the compressed air is admitted to, and shut off from, the engine cylinder, is worked by hand. Frequently but one impulse with compressed air is needed, and seldom more than three.

One air-reservoir of 90 cu. ft. capacity is used, which is kept full of air at about 90 lb. pressure by the intermittent use of a motor-driven air-compressor, of 8-in. diameter by 6-in. stroke, running at about 60 revolutions per minute. A steam-driven locomotive air-pump was used in starting up the plant, and, supplied by the gas-plant boiler, it remains as a stand-by to the compressor.

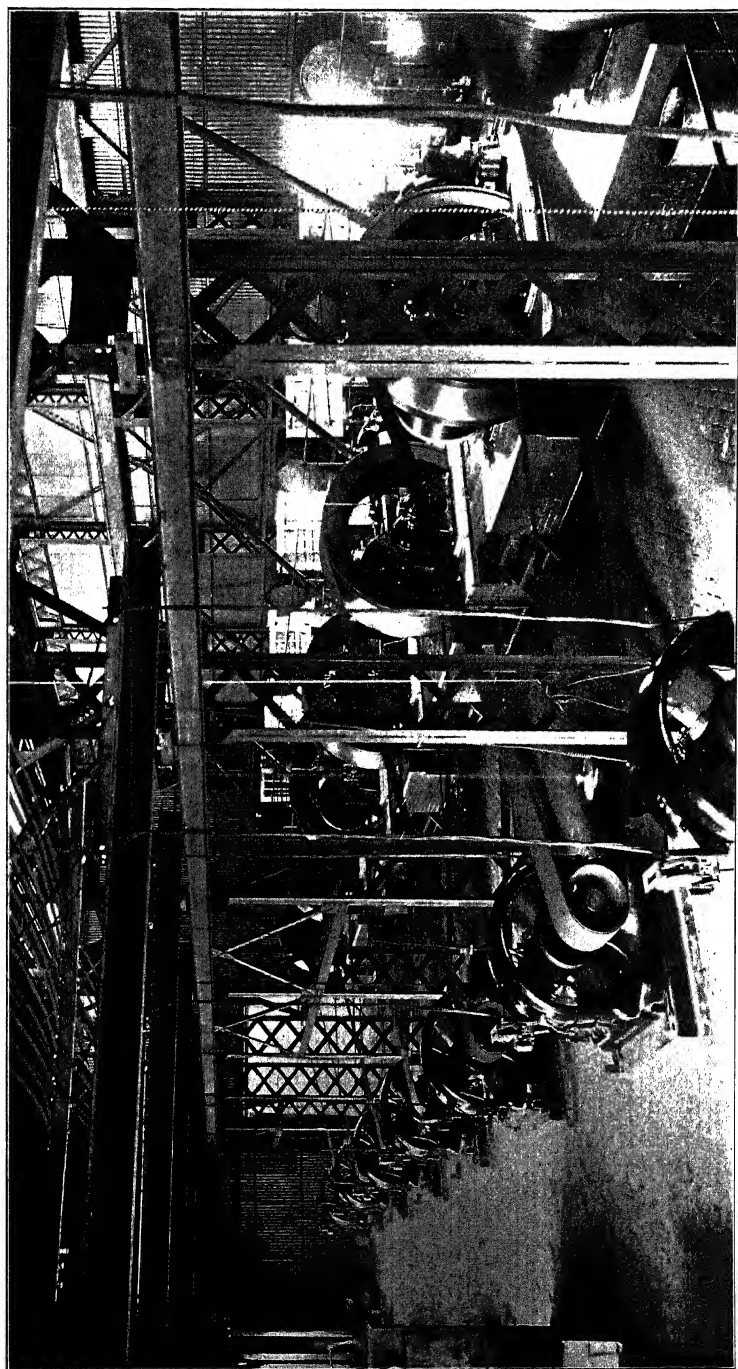
Opposite each engine is a tank 11 ft. by 18 ft. by 3 ft. deep, which is placed outside of the power-house on a level 20 ft. above the engine. Pipes from the inlet and outlet of the engine water-jacket are connected to the bottom of the tank; and the heat of the cylinder automatically keeps the jacket-water in circulation. The flat form of tank gives sufficient surface to cool the water by radiation and evaporation.

Although this method is automatic and requires no power, it is questionable whether it would not be better to admit the water positively from a storage tank and expend power to pump back to the tank over a cooling-tower; since a definite control of the jacket-water is useful when the engines are being driven hard. The question is one for decision according to the conditions of each case.

The water at Nacozari is very hard, and the formation of scale in the water-jackets of the engines caused some trouble at first. The water needed to make up for evaporation is now purified by treatment with soda. It flows from the mixing-tank to a deep conical-bottom tank, of the type used in the mill for thickening pulp. The sludge of lime and magnesia precipitates escapes at the bottom outlet, while the clarified water, overflowing the rim, passes to the jacket-water tanks.

The main bearings of the engines are oiled continuously by an automatic oil-circulating and filtering system. Two men on a shift are required for attendance, an engineer and a helper.

FIG. 3.



INTERIOR VIEW OF THE POWER-HOUSE AT NACOZARI.

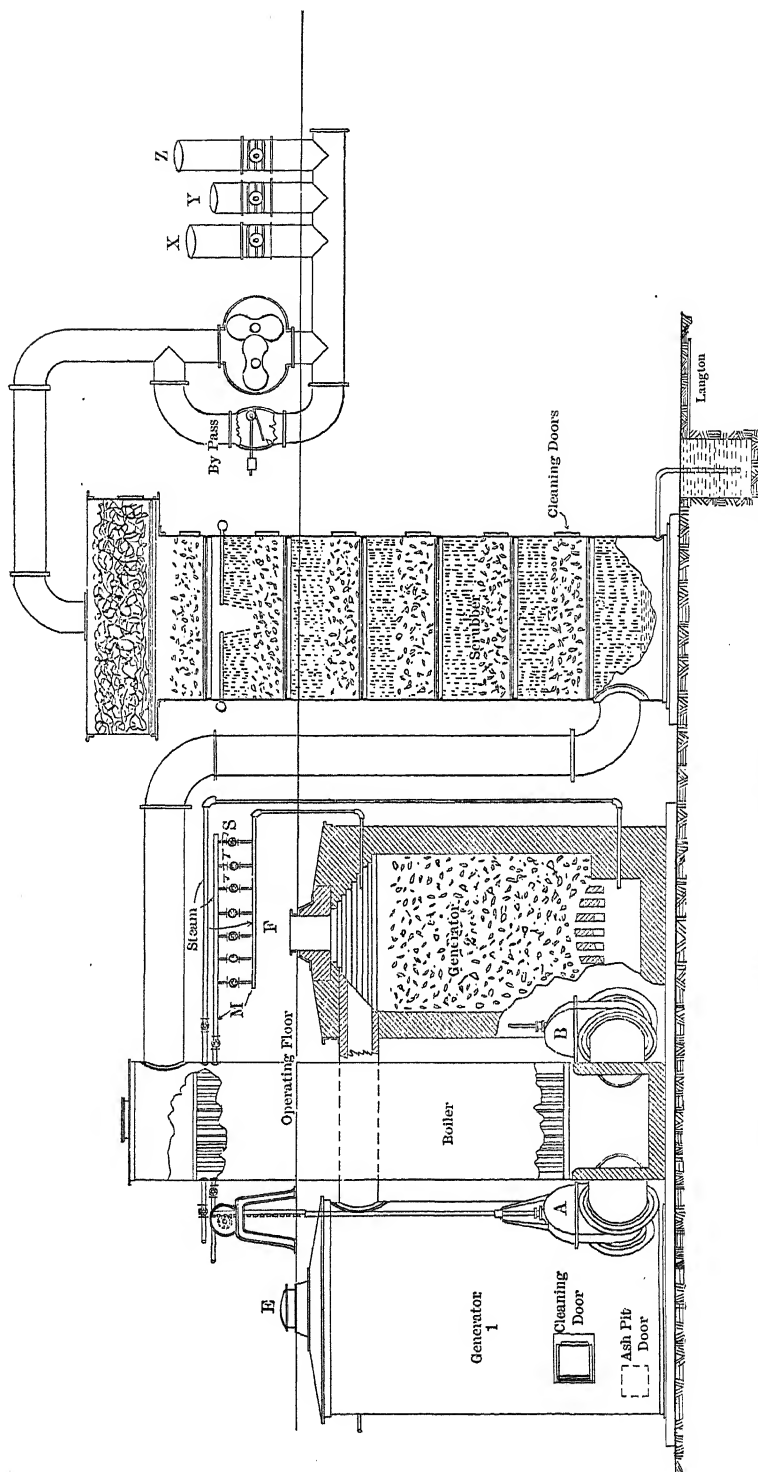
## DESCRIPTION OF GAS-HOUSE.

The gas-producers are those of the Loomis-Pettibone Gas Machinery Co., arranged as shown in Figs. 4 and 5. The draught through the producers is created by the suction of the gas-exhauster, and air enters through the open feed-doors in the top of the producers. Through the same doors the fuel is charged by hand. The air thus comes into contact first with the fresh fuel, and the products of combustion, together with any distillates from the fresh fuel, are drawn downwards through the deep charge of incandescent fuel and out through the fire-brick grate at the bottom of the producer. The high temperature of the gas drawn out of the producer is partially utilized to generate steam in the boiler, through which the gas next passes, and from which it emerges cooled to about the ordinary temperature of chimney gases. It is further cooled to about atmospheric temperature in passing through the water-sprayed coke-filled trays of the scrubber, from which it finally reaches the exhauster which delivers it into the holders.

The quantity of gas made is controlled by the speed of the exhauster, which is of the positive type. The vacuum on the suction side of the exhauster, required to produce the necessary draught, depends upon the condition of the fires and the quantity of gas being made, but ordinarily it is from 12 to 24 in. of water. The exhauster delivers against the 6 in. of water-pressure to which the gas holders are weighted. The total difference of pressure, therefore, is from 11 to 18 oz.

The steam raised in the boiler is used intermittently to make water-gas. For this purpose the feed-doors E and F (Fig. 4) are closed, and also the valve B. Steam is then admitted under the grate of producer No. 2, and is dissociated at the temperature of the fire, forming water-gas in its passage up through producer No. 2, across the top connecting-pipe to producer No. 1, down through the latter, and out by valve A to the boiler, the scrubber and the gas-holder. The next time water-gas is made the direction is reversed; the valve A is closed, and the gas-run is made up through producer No. 1 and down through No. 2. Or, when half the run is made up producer No. 1, steam may be shut off while valves A and B are reversed, and the other half of the run made up producer No. 2.

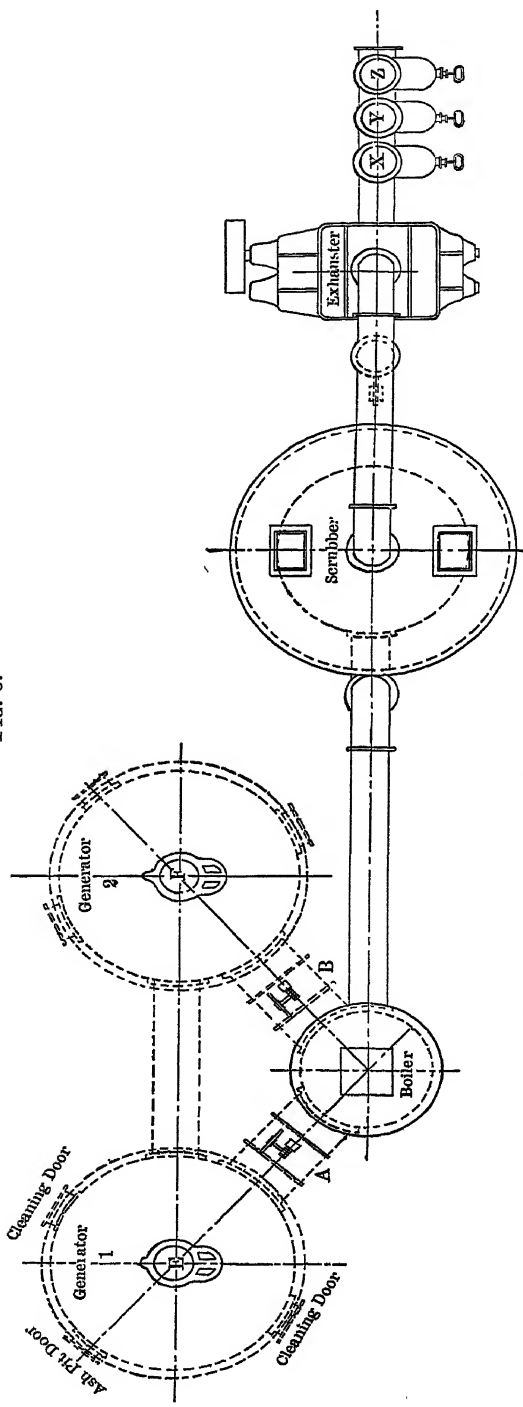
FIG. 4.



LONGITUDINAL SECTION OF THE LOOMIS-PETTIBONE GAS-PRODUCER AT NACOZARI.



Fig. 5.



PLAN OF THE LOOMIS-PETTIBONE GAS-PRODUCER AT NACOZARI.

The latter plan is generally followed at Nacozari, where the method adopted for making gas requires only short and infrequent runs of water-gas.

A gas-holder of 15,000 cu. ft. capacity receives the gas which is to be used in the engines, and gives the necessary elastic storage between the production in the gas-house and the consumption in the power-house. A second gas holder of 5,000 cu. ft. capacity is used to receive, and to keep separate, the water-gas made by the intermittent runs. By means of valves X, Y and Z, the gas is delivered to either of the holders, or is allowed to escape into the air through the purge pipe.

The producers have an inside diameter of 6 ft. 9 in., and, when charged with coal to the highest level practicable, hold a fuel-charge 8 ft. deep. A generating-set consists of two producers and one boiler. At Nacozari there are two generating-sets which deliver to a common scrubber and a common exhauster, with space provided for the addition of a second scrubber and a second exhauster if desired in the future. The sets are alternated in use. One is run till the accumulation of ash or clinker begins to limit its capacity; the other set is then fired, and blown to incandescence by a centrifugal-exhaust fan. The valve connecting the new set to the scrubber and exhauster is opened, and the valve connecting the old set is closed. There is no interruption to the process of gas-making. By means of the cleaning-doors at the grate-level, the producers of the old set are then emptied, and cleaned of ash and clinker. The cleaning-doors are closed and sealed air-tight, and the producers charged with fuel ready to fire up again when their turn comes. The length of run on one set depends on the rate of gas-making and on the percentage of ash in the fuel. At the rate of gas-production required for an average load of 600 brake-h.p., about 80 tons of the ordinary western coal, or about the same weight of wood, may be gasified before the accumulation of ash and clinker necessitates cleaning.

The water used in the scrubber is caught in a lower tank, and pumped back over a small cooling-tower to the head tank supplying the scrubber. The feed-water for the boiler is drawn from the supply already purified by soda-treatment for use in the engine-jackets.

The power required to drive the pump and the exhauster in

the gas-house is furnished by an electric motor. In case the whole plant is shut down, the holder is left full of gas, which gives sufficient power to start the gas-plant and to run it until it is again producing gas. A small steam-engine was used to start the plant first, when there was no gas in the holder, and remains in reserve for a similar emergency.

The plant has been operated with wood and with coal, both anthracite and bituminous. The down-draught principle of the producers, by which all gases pass out through the bottom of the fire, has proved thoroughly efficacious in producing fixed gases from all kinds of fuel used at Nacozari. The water from the scrubber rarely shows even that trace of tar which manifests itself by an iridescent film on the surface of the water in the lower scrubber tank.

A considerable quantity of very finely divided carbon is carried through the scrubber and into the holder and mains. It is unnoticeable at the engine valves, but it gradually deposits in the gas mains, which must occasionally be flushed out. An efficient dry scrubber to prevent its passage would be an improvement.

With fuel delivered in the gas-house bins, the attendance required is one gas maker and one helper on each shift, and extra labor for cleaning the producers.

#### METHOD OF MAKING GAS.

In the Loomis-Pettibone producers, as ordinarily used with the object of making as much water-gas as possible, the aim is to get the fires very hot before making water-gas, and to continue the admission of steam until the fires are cooled to a point where a considerable portion of the steam passes through unaltered; the making of producer-gas with air is then resumed, with the aim of quickly blowing the fires as hot as possible, preparatory to steaming them again to make water-gas. This procedure necessarily causes great variation in the temperature of the fires, and this, though it has little effect on the quality of the water-gas, has a very great effect on the quality of the producer-gas, since the latter, when made by blowing a cold fire, has a large percentage of  $\text{CO}_2$  and is lean in the combustible  $\text{CO}$ ; while with a hot fire, the percentage of  $\text{CO}_2$  is very much less, and that of  $\text{CO}$  more than correspondingly in-

creased,—one volume of  $\text{CO}_2$  being converted into two volumes of  $\text{CO}$ .

The original intention was to make producer-gas and water-gas at Nacozari in the usual manner by alternate blowing and steaming, the two kinds of gas to be delivered into two separate holders, and from these drawn in definite proportions through a mixing-valve into the gas-main leading to the engines. In a large plant, making gas from several sets of producers simultaneously, this mode of operation would doubtless deliver to the engines a sufficiently uniform quality of mixed gas, because the variation in quality of producer-gas from one set would be averaged in the holder by the gases from the other sets, which would never be simultaneously at the same stage of temperature; but in a plant operating only one set of producers, as at Nacozari, the quality of the mixed gas was too variable. The method of gas-making was therefore modified to avoid great variations in the temperature of the fires. The steam-pipe connections, marked M in Fig. 4, permit the introduction of steam together with the air to the top of the fire, and by this means the excess heat of the fire caused by the air-blowing may be concurrently used to make water-gas; so that by properly proportioning the quantities of steam and air introduced, the fires may be maintained at a practically uniform temperature for a long period of time. As a matter of fact, at Morenci, Ariz., where the Detroit Copper Mining Co. operates an identical gas-plant, at times when the load was very uniform, runs have been made in this manner without touching a valve for 24 hours. Obviously, however, such a state of equilibrium requires an adjustment of air and steam, rather too fine for easy every-day working. The method preferred, therefore, is to admit somewhat less top steam than could be used, in order to allow the fire to increase slowly in temperature; and then, without allowing the temperature to become excessive, to make a short water-gas run which only slightly cools the fire, and to resume immediately the making of mixed gas with top steam. By this method the fire is never allowed to become cold enough to make an undue proportion of  $\text{CO}_2$ , and its maximum and minimum temperatures vary but slightly from an average high temperature.

In addition to the fact that this method obviates the neces-

sity of nice adjustment, the occasional water-gas runs (alternately up and down the producers as previously described in this paper) have a very beneficial effect upon the fire. They distribute the heat more uniformly throughout the column of fuel, and in a manner shake it up and thus prevent the formation of holes or masses of clinker.

The water-gas made separately by this method and delivered into the water-gas holder is gradually fed from the latter into the main holder which supplies the gas-engines, the water-gas holder being weighted to give an inch or two greater pressure than that of the main holder. The rate of feed is controlled by a throttle-valve, in a pipe connecting the two holders, which the operator manipulates so that the rate of feeding water-gas approximately equals the rate of its production. No particular accuracy is required, nor are pains taken in regulating the quantity of water-gas made separately. The operator is governed principally by the temperature of the fire, as indicated by its appearance and by the rise of steam-pressure in the boiler. If the fall of the water-gas holder necessitates the making of water-gas before the fires seem quite hot enough, the throttle-valve opening is slightly lessened; and vice versa. This is not a point which varies much during the day, or even from day to day, further than that, at the beginning of a run on a new set of generators, the production of water-gas may differ from the quantity made several days later at the conclusion of the run on the same set.

By this method it has been found quite easy to make gas, such that the variations in its quality, shown by analyses and calorimetric readings, have no observable effect upon the working or capacity of the engines.

The regulating cocks, marked S in Fig. 4, are drilled with small holes, and the quantity of the top steam is regulated by the number of cocks open. This gives a graduated regulation and the ability to duplicate conditions exactly. The quantity of air is independently controlled by the speed of the exhauster, or by slightly opening, to a graduated extent, the by-pass valve of the exhauster.

Gas for power must be of uniform quality; and a plant whose product is used for power alone must make uniform gas at varying rates. To do this with greatest economy, all the gas

made should be of the same quality, that there may be no need of wasting lean gas through the purge-valve. The method used at Nacozari seems well adapted to meet these conditions. The fall of the main holder during the 3 or 4 minutes occupied in putting water-gas into the separate holder is compensated by its rise during the interval between water-gas runs. The production is set sufficiently in excess of the consumption to effect this compensation; and under a steady load, the position of the holder ranges between the same index-marks with great regularity. But with a change in the power-demand lasting for some time, the rise of the holder ceases to equal its fall. This discrepancy is met by changing the quantities of air and steam without altering their ratio, and by a corresponding change in the interval between the water-gas runs. The result is that the time occupied by one cycle of operations is shortened for an increased, and lengthened for a decreased, demand. Nevertheless, coal, air and steam are consumed in the same ratio as before, and in constant quantities per cycle. Consequently, the range of temperature in the fire remains unchanged, and the proportion of top-steam gas to separate water-gas is approximately the same. This method therefore gives, at all rates of production, those nearly uniform conditions in the fire which are of the first importance for making gas of uniform quality.

#### GAS MADE FROM WOOD.

The most novel feature of the Nacozari power-plant is the use of producer-gas made from wood alone. No guiding experience was found for this process; but, with the desire to utilize as far as possible the limited local wood-supply, the gas-producer plant was selected with the object, among other things, of its being suitable to make the attempt to use, if not wood alone, at least a considerable admixture of wood with bituminous coal. The power-plant was started in operation July 31, 1900, with coal; but for some months there was no convenient opportunity to experiment with wood, and it was not until February 4, 1901, that the first trial of wood alone was made.

The most obvious difficulty to be feared arose from the large proportion of condensible distillates yielded by wood, and the danger that some portion of these might be imperfectly fixed

in passing through the producer. The trouble from tar deposited in the gas apparatus and pipes would be serious, and even a small quantity of tar in the gas itself is a fertile source of trouble at the engine valves. Unless a permanent gas could be made from wood, this fuel would be unavailable.

The first care, therefore, was to insure that there should be a bed of charcoal on the grate sufficient to form an adequate fixing zone. To obtain this the producers were filled about 5 ft. deep with cordwood sawn in blocks about 6 in. long, and the contents blown with a slow fire for four or five hours before the gas was turned into the holder. The gas, as it proved, was turned into the holder too soon. At first, it contained some tar; and it was not until after about three hours' operation that the charcoal accumulated in sufficient quantity, so that the producers delivered fixed permanent gases to the holder. This was evident on exposing white paper to the gas, and is also clearly indicated in the series of calorimetric readings (Table I.), which were begun one hour after the gas was turned into the holder. The high calorific values of the first readings, gradually diminishing to the nearly uniform values which correspond with the results subsequently obtained, mark the disappearance of partially condensible distillates.

TABLE I.—*Calorimeter-Readings of Wood-Gas Made Feb. 4, 1901. Reduced to Sea-Level and 60° F.*

Time.	Calorific Power.
	B. T. U.
9.00 P.M.	174.5
9.25 P.M.	158.3
9.40 P.M.	162.0
10.05 P.M.	151.2
10.40 P.M.	142.4
10.55 P.M.	134.8
11.10 P.M.	139.8
11.20 P.M.	137.0
11.40 P.M.	125.4
12.05 A.M.	134.0

This preliminary trial lasted more than twelve hours, until the supply of sawn wood was exhausted. As was expected, upon drawing the fires a considerable quantity of charcoal was obtained, which was available for use as a bed in starting the

succeeding fires. But before the next trial was made it was suggested that for this purpose coke would be better than charcoal. Although in the first trial charcoal in sufficient quantity had proved effective in fixing all of the gases, it was felt, rightly or wrongly, that the coke would be a stouter and more reliable screen than the comparatively frail charcoal; and in particular, that, as a reinforcement to the charcoal formed above it by the action of the fire, it would effectually safeguard against the possible danger that temporary holes might be burned through the charcoal, by which tar-vapors might escape from the producer, but for the fixing-action of the more enduring coke. However correct this opinion might be, a coke-bed would certainly be an additional precaution against the serious trouble which might result from delivering imperfectly-fixed gases to the holder, and would give to the operator great relief from anxiety on this score. It was therefore thought best without further experiment to use coke, which was available at Nacozari; and the practice has been continued ever since. The experience at Nacozari, therefore, does not determine whether it would be safe to dispense with the use of coke.

The second trial, beginning March 14, was carried on for 66 hours with perfect success; and after a third trial of 36 hours' duration, made a few days later, had proved that there was no difficulty in repeating the previous excellent results, measures were taken to gather a supply of wood sufficient to operate the plant with wood-gas exclusively.

The analyses and calorific values of wood-gas made during the second and third trials are given in Table V.

The wood used for the first trial was sawn needlessly small. For the second trial the bulk of the wood was in half lengths of cordwood, with about 10 per cent. whole lengths and 10 per cent. short blocks. The producers at Nacozari are too small to use whole-length cordwood alone. Half lengths are satisfactory for fuel, but small sticks may be quickly sawn into 6-in. lengths, giving a stock of small blocks for filling holes or quickly building up the fires. The wood is cut for packing on burros, and is little more than 3 ft. long; there is also a large proportion of crooked sticks. As actually piled in the yard and paid for, it weighs about 2,200 lb. per cord. A motor-



driven, semi-portable circular-saw frame, which may be shifted along the wood-pile, will handle 14 cords in 10 hours, cutting the stick once, and the small stuff into short blocks. This is equivalent to more than 3,000 lb. per hour.

The wood used is principally scrub white-oak, with about 10 per cent. of ash. The quantity of moisture contained is best shown by the series (Table II.) of moisture-samples taken from wood actually used for making gas at Nacozari in the autumn of 1902:

TABLE II.—*Water-Content of Wood Used at Nacozari in 1902.*

Date.	Per Cent.	Date.	Per Cent.	Date.	Per Cent.
Oct. 22	12.94	Nov. 1	23.42	Nov. 13	21.60
Oct. 23	17.30	Nov. 2	20.97	Nov. 14	23.97
Oct. 24	16.31	Nov. 4	22.92	Nov. 15	25.21
Oct. 25	17.43	Nov. 5	22.19	Nov. 17	9.67
Oct. 27	19.68	Nov. 6	23.25	Nov. 18	22.33
Oct. 28	15.23	Nov. 7	24.24	Nov. 20	18.60
Oct. 29	16.86	Nov. 8	21.48	Nov. 24	16.00
Oct. 31	21.00	Nov. 10	22.08	Average,	19.76

The sample which gave 9.67 per cent. of water was taken from cordwood seasoned for two years in the open air.

Ordinarily, no top steam is used in making wood-gas, the contained moisture taking its place. Drier wood, without top steam, means somewhat more frequent water-gas runs. With very wet wood the fires do not become hot enough to make good gas. About 3 ft. depth of coke is put in to form the precautionary bed, the greater part of which is recovered.

#### QUALITY OF GAS PRODUCED.

The character of the gas produced from different fuels is shown by the series of analyses (Tables III. to VI., inclusive), made by A. S  ndberg, Ph.D., of Lund, during the final trials of the gas-making plant, which extended from February 16 to March 20, 1901.

TABLE III.—*Analyses of Gas from Cerrillos Anthracite Coal.*

Date.	Components.							Calorific Value.
1901.	CO.	H.	CH <sub>4</sub> .	C <sub>2</sub> H <sub>2</sub> n.	CO <sub>2</sub> .	O.	N.	B. T. U. Per Cu. Ft. at 60° Fahr. and 29.9 In. Barometric Pressure.
	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	
Feb. 16.	23.20	13.43	2.13	0.21	8.2	0.0	52.84	141.7
Feb. 19.	21.55	14.50	2.18	0.20	8.2	0.25	53.18	140.5
Feb. 23.	18.9	12.78	2.07	0.10	10.4	0.0	55.81	122.9
Feb. 27.	20.8	11.47	2.78	0.10	8.8	0.1	55.95	132.6
Mar. 12.	20.9	10.40	2.31	0.20	7.3	0.5	58.39	126.4
Mar. 12.	24.4	9.90	1.10	0.10	5.1	0.4	59.00	122.4
Mar. 17.	20.5	14.24	2.12	0.30	9.2	0.0	53.64	137.2
Mar. 18.	22.7	12.46	1.65	0.20	7.4	0.2	55.39	132.2
Mar. 18.	22.4	13.45	1.14	0.20	8.6	0.4	53.81	129.3
Mar. 20.	20.6	14.18	1.15	0.20	7.8	0.2	55.81	125.9
Average.	21.6	12.68	1.86	0.18	8.1	0.2	55.38	131.1

TABLE IV.—*Analyses of Gas from Cerrillos Bituminous Coal.*

Date.	Components.							Calorific Value.
1901.	CO.	H.	CH <sub>4</sub> .	C <sub>2</sub> H <sub>2</sub> n.	CO <sub>2</sub> .	O.	N.	B. T. U. Per Cu. Ft. at 60° Fahr. and 29.9 In. Barometric Pressure.
	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	Vol- ume Per Cent.	
Mar. 2...	20.3	14.43	2.28	0.2	8.2	0.0	54.59	137.1
Mar. 3...	22.2	11.58	1.74	0.2	7.4	0.2	56.68	128.7
Mar. 4...	18.4	12.42	2.11	0.2	7.8	0.0	59.17	122.8
Mar. 5...	21.6	12.99	2.78	0.2	6.5	0.0	55.93	141.7
Mar. 6...	19.5	13.21	3.01	0.2	7.7	0.0	56.38	136.37
Mar. 7...	19.9	13.84	2.30	0.2	8.4	0.0	55.36	134.18
Average.	20.32	13.08	2.37	0.2	7.66	0.04	56.35	133.47

TABLE V.—*Analyses of Gas from Wood.*

Date.	Components.							Calorific Value.
1901.	CO.	H.	CH <sub>4</sub> .	C <sub>2</sub> H <sub>2n</sub> .	CO <sub>2</sub> .	O.	N.	B. T. U. Per Cu. Ft. at 60° Fahr. and 29.9 In. Barometric Pressure.
	Volume Per Cent.	Volume Per Cent.	Volume Per Cent.	Volume Per Cent.	Volume Per Cent.	Volume Per Cent.	Volume Per Cent.	
Mar. 14..	11.0	18.91	3.10	0.3	17.9	0.0	48.79	131.5
Mar. 15..	14.8	20.61	1.67	0.3	14.7	0.0	47.92	134.8
Mar. 15..	11.8	22.31	3.10	0.4	17.2	0.2	45.19	146.55
Mar. 16..	14.0	22.35	2.45	0.2	15.2	0.0	45.80	144.0
Mar. 16..	12.5	22.35	3.06	0.3	16.4	0.1	45.29	146.9
Mar. 19..	15.5	19.28	2.31	0.2	14.4	0.4	47.91	137.6
Average.	13.27	20.97	2.61	0.28	15.96	0.11	46.82	140.22

TABLE VI.—*Type Composition of Coal-Gas and Wood-Gas.*

Components.	Coal-Gas.	Wood-Gas.
	Volume Per Cent.	Volume Per Cent.
CO.....	20.0	14.0
H.....	14.0	20.0
CH <sub>4</sub> .....	2.0	2.0
C <sub>2</sub> H <sub>2n</sub> .....	0.2	0.2
CO <sub>2</sub> .....	8.2	16.0
O.....	0.1	0.1
N.....	55.5	47.7
Total.....	100.0	100.0
Calorific value per cu. ft., calculated at sea-level and at 60° F.	B. T. U. 132.	B. T. U. 132.

Tables III, IV, and V. show that the composition of producer-gas made from bituminous coal is practically the same as that made from anthracite coal, but differs radically from gas made from wood; the calorific values of all three gases, however, are nearly identical. The type composition of coal-gas and wood-gas is given in Table VI.

As compared with coal-gas, the proportions of CO and H in wood-gas are reversed, and the percentage of CO<sub>2</sub> is doubled. With the typical compositions given above, the wood-gas shows less tendency than the coal-gas to pre-ignition in the engines. This effect appears to be due to the large proportion of CO<sub>2</sub>, but the reason why this is so is not apparent. Other experience

at Nacozari has certainly shown a very marked tendency to pre-ignition in the engines when the gas is approximately of the following composition by volume per cent. :—

CO, 14; H, 20; CH<sub>4</sub>, 2; C<sub>n</sub>H<sub>2n</sub>, 0.2; CO<sub>2</sub>, 8.2; O, 0.1;  
and N, 55.5; total, 100  
(B. T. U. per cu. ft. = 132),

and yet this gas had the same ratio of incombustibles to combustibles, and the same composition of the latter, as the cool-working wood-gas. The only difference is in the composition of the incombustibles; and it is very obscure why a change in the relative proportions of CO<sub>2</sub> and N should produce such markedly different results in the behavior of the engines.

#### QUANTITATIVE RESULTS.

The results obtained from the Nacozari power-plant are best shown by the extracts given in Tables VII. to X., inclusive, from a very complete record made, during an entire run on one generating-set, by Dr. A. Sandberg, metallurgist of the works, and Mr. George M. Douglas, engineer in charge of the power-plant.

TABLE VII.—*Record of Six Days' Run on One Generating-Set at the Nacozari Gas- and Power-Plant, from Nov. 21 to 27, 1902.*

#### MEASUREMENTS OF POWER.

Date.	Hours. Run.	Average Readings at Switch-Board.			Equiv. B.H.P., Allowing 90 Per Cent. Efficiency in Generator.	In Gas-House.	
		Amperes.	Volts.	E.H.P.		Average E.H.P. Used.	Vacuum at Exhauster in Inches of Water.
Nov. 21..	14	1380	250	464	515	18.1	14
Nov. 22..	18	1274	250	428	475	14.9	20
Nov. 23..	24	1398	250	470	522	15.9	24
Nov. 24..	24	1210	250	404	448	17.3	26
Nov. 25..	24	1330	250	446	495	18.0	27
Nov. 26..	24	1285	250	431	474	18.7	28
Nov. 27..	10	1328	250	445	494	19.1	31
Total.....	138	.....	.....	.....	.....	.....	.....
Average.....		1315	250	441.7	489	16.7	24.3

NOTE.—Electrical quantities were obtained by the average of periodical readings.

TABLE VIII.—*Record of Six Days' Run on One Generating Set at the Nacozari Gas- and Power-Plant, from Nov. 21 to 27, 1902.*

## RUNNING TIME OF ENGINES.

Engine.	Actual Number of Hours.	Per Cent. of Total Time.
No. 1.....	124	90.0
No. 2.....	137	99.2
No. 3.....	121	87.6
No. 4.....	136	98.5
No. 5.....	132	95.6
No. 6.....	138	100.0
No. 7.....	101	73.2
No. 8.....	123	89.1
Average.	126.5	91.6

TABLE IX.—*Record of Six Days' Run on One Generating Set at the Nacozari Gas- and Power-Plant, from Nov. 21 to 27, 1902.*

## ANALYSES OF GAS MADE.

Date.	Gas Supplied to Engines.							B. T. U. Per Cu. Ft.	
	CO.	H.	CH <sub>4</sub> .	C <sub>2</sub> H <sub>2n</sub> .	CO <sub>2</sub> .	O.	N.	Reduced to 60° F. and 29.9 in.	Actual, at 80° F. and 26.4 in.
Nov. 22..	13.40	17.70	3.32	0.20	15.00	0.2	50.18	135.92	115.4
Nov. 23..	14.00	19.50	2.75	0.20	15.80	0.4	47.35	137.90	117.0
Nov. 24..	14.80	20.03	2.62	0.40	14.00	0.2	47.95	144.05	122.2
Nov. 25..	14.50	19.97	1.41	0.30	15.60	0.2	48.02	129.10	108.8
Nov. 26..	13.30	19.41	2.81	0.50	15.00	0.7	48.28	141.78	120.2
Nov. 27..	13.20	18.00	1.46	0.50	14.10	0.4	52.34	122.44	104.0
Average	13.87	19.10	2.39	0.35	14.91	0.35	49.02	135.20	114.6
Water-Gas.									
Nov. 24..	8.90	55.64	3.37	0.40	21.00	0.4	10.49	246.62	
Nov. 25..	10.40	53.80	3.30	0.50	22.40	0.3	9.30	246.44	
Nov. 27..	10.30	53.06	3.69	0.50	20.60	0.1	11.75	247.58	
Average.	9.87	54.17	3.45	0.47	21.33	0.27	10.51	246.88	

NOTE.—Heat units are calculated on the basis of:—

CO = 320 B. T. U. per cu. ft. at 60° F. and 29.9 in. barometer.

H = 320 B. T. U. per cu. ft. at 60° F. and 29.9 in. barometer.

CH<sub>4</sub> = 1,000 B. T. U. per cu. ft. at 60° F. and 29.9 in. barometer.C<sub>2</sub>H<sub>2n</sub> = 1,600 B. T. U. per cu. ft. at 60° F. and 29.9 in. barometer.

TABLE X.—*Record of Six Days' Run on One Generating-Set at the Nacozari Gas- and Power-Plant, from Nov. 21 to 27, 1902.*

## MISCELLANEOUS QUANTITIES AND RESULTS.

*Oil:*

Total quantity used in engine cylinders, . . .	24 gallons.
Total quantity used in engine bearings, . . .	22.5 gallons.

*Fuel:*

Coke put in generators, . . . . .	5,740 lb.
Coke taken from generators fit for use again, . . .	4,642 lb.
Total coke consumed, . . . . .	1,088 lb.
<sup>a</sup> Wood used until fires were ready, . . . . .	10,000 lb.
<sup>a</sup> Wood used while making gas, . . . . .	190,750 lb.
<sup>a</sup> Total wood consumed, . . . . .	200,750 lb.
Total B.H.P.-hours, . . . . .	67,482
<sup>a</sup> Wood consumed per B.H.P.-hour, . . . . .	2.97 lb.

*Gas:*

Total number of water-gas runs made, . . . . .	208
Average interval between runs, . . . . .	40 min.

*Steam (approximately estimated) :*

Used in making water-gas (at 125 lb. pressure), . .	66,000 lb.
Blown away at safety valve (at 130 lb. pressure), .	14,500 lb.
Water blown from boiler at blow-off cock, . . .	3,500 lb.
Total feed-water, . . . . .	84,000 lb.
Average temperature of feed-water, . . . . .	78° F.

*Scrubber-Water per Hour (approximate measurements) :*

Average quantity passed through scrubber, . . .	3,800 gallons.
Average inlet-temperature, . . . . .	72.5° F.
Average outlet-temperature, . . . . .	107.6° F.
Sensible heat extracted in scrubber per hour, . .	1,111,500 B. T. U.

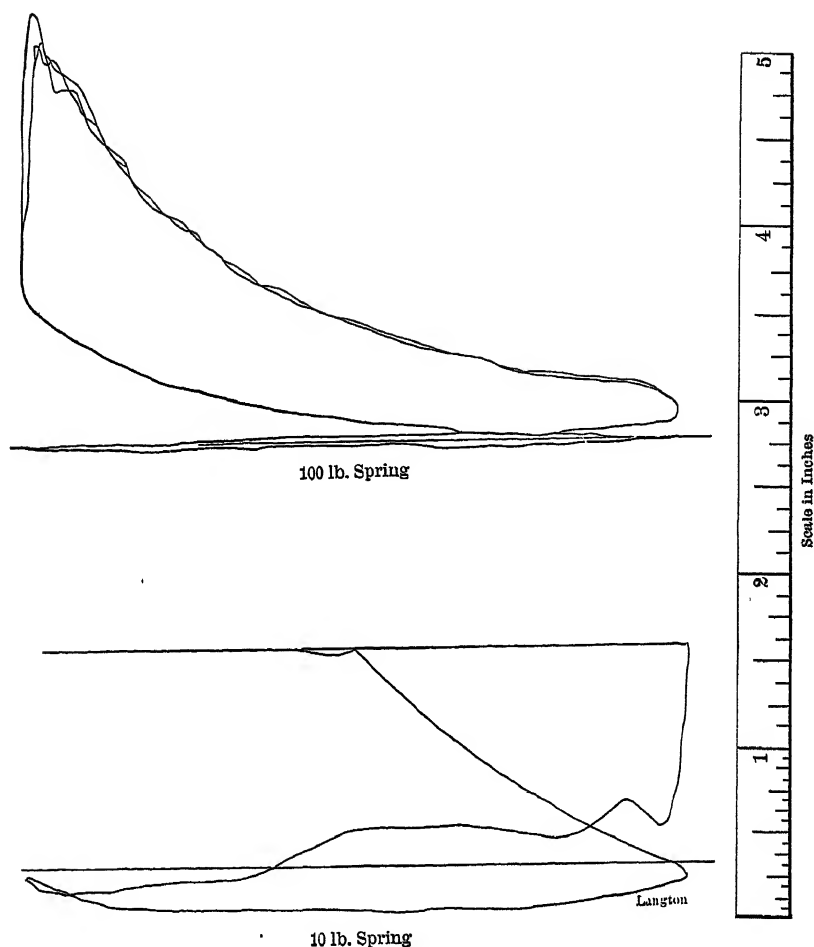
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<sup>a</sup> Weights of wood are commercial figures, including shrinkage, charged against the plant on the books.

## RECORDS OF THE GAS-ENGINES.

Copies of some of the indicator-diagrams, taken during the six days' run from three gas-engines of the power-plant, at Nacozari, are shown in Figs. 6, 7 and 8.

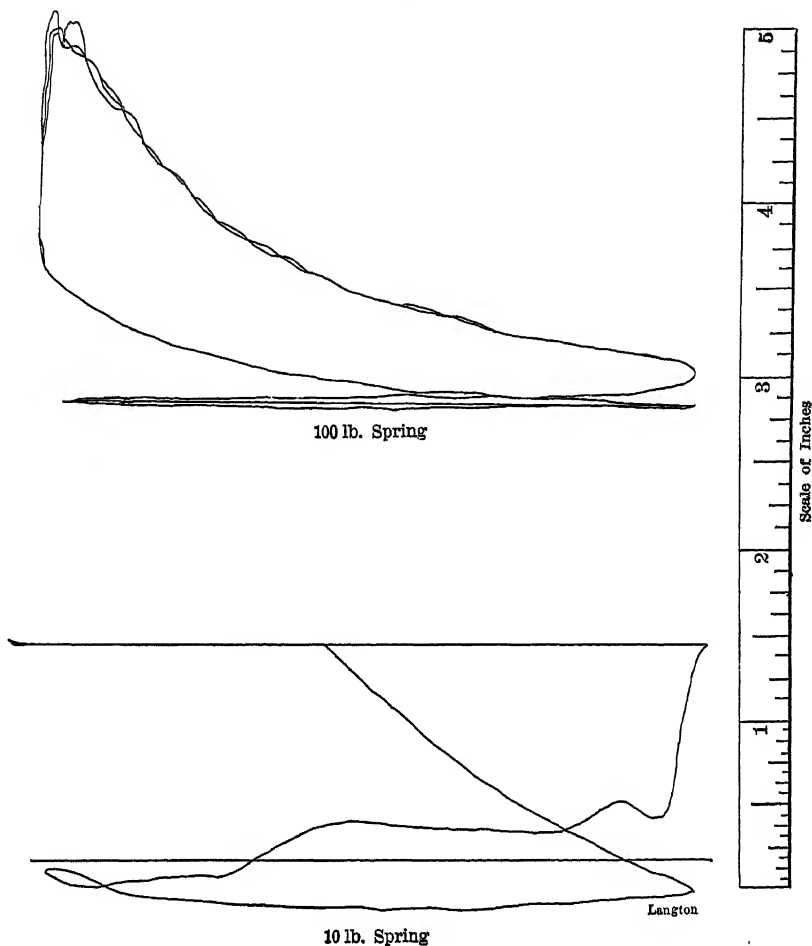
FIG. 6.



18.5 in. by 24 in. cylinder, . . . . . 200 R. P. M.  
 Gas, 115 B. T. U., actual calorimeter reading.  
 Current, . . . . . 230 amperes and 250 volts.  
 M.E.P., . . . . . 62.8 lb.  
 I.H.P., . . . . . 103.4  
 E.H.P., . . . . . 77  
 Combined efficiency, . . . . . 74.5 per cent.  
 Allowing 90 per cent. efficiency from flywheel to switch-  
 board,  
 B.H.P. would be, . . . . . 85.6  
 And mechanical efficiency would be, . . . . . 82.8 per cent.

INDICATOR DIAGRAM TAKEN NOVEMBER 23, 1902, FROM ENGINE No. 3, OF  
 THE POWER-HOUSE AT NACOAARI.

FIG. 7.

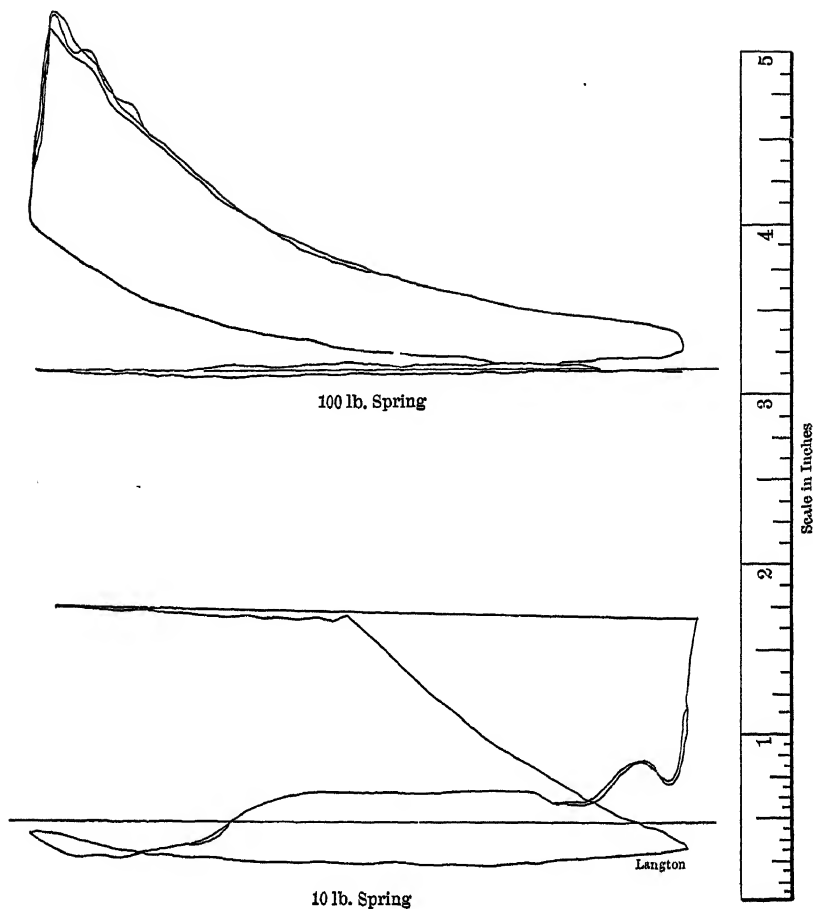


18.5 in. by 24 in. cylinder,	200 R. P. M.
Gas, 113 B. T. U., actual calorimeter reading.	
Current,	215 amperes and 250 volts.
M.E.P.,	59 lb.
I.H.P.,	97.3
E.H.P.,	72
Combined efficiency,	74 per cent.
Allowing 90 per cent. efficiency from flywheel to switch-board,	
B.H.P. would be,	80
And mechanical efficiency would be,	82.2 per cent.

INDICATOR DIAGRAM TAKEN NOVEMBER 24, 1902, FROM ENGINE NO. 5, OF  
THE POWER-HOUSE AT NACOAARI.



FIG. 8.



18.5 in. by 25 in. cylinder, . . . . . 200 R. P. M.

Gas, 122 B. T. U., actual calorimeter reading.

Current, . . . . . 200 amperes and 250 volts.

M.E.P., . . . . . 55.8 lb.

I.H.P., . . . . . 92

E.H.P., . . . . . 67

Combined efficiency, . . . . . 72.8 per cent.

Allowing 90 per cent. efficiency from flywheel to switch-board,

B.H.P. would be, . . . . . 74.4

And mechanical efficiency would be, . . . . . 80.9 per cent.

INDICATOR DIAGRAM TAKEN NOVEMBER 26, 1902, FROM ENGINE NO. 8, OF  
THE POWER-HOUSE AT NACOAARI.

## REMARKS ON THE USE OF GAS-ENGINES.

The sustained capacity and satisfactory working of a gas-engine depend on its being kept in thoroughly good condition. The principal meaning of "good condition" is, that the valves should seat freely, and be tight when seated; and that the inside of the cylinder should not be foul with carbonized oil. Leaky or sticky valves and dirty cylinders are the causes of nine-tenths of the ills peculiar to gas-engines. A gas-valve sticking open may cause the engine to run away; but, with this one exception, every other trouble manifests itself by reducing the capacity of the engine, or by bringing it to an actual stop. Hence good condition is essential to satisfactory operation. It is easily maintained by periodical attention—cleaning the cylinders, slightly regrinding the valves to their seats, and a care of the igniters, which differs according to the type of igniter used.

This general examination should be made methodically and at regular periods, after the engine has been running a certain number of hours, as determined by experience in each plant. It cannot be deferred indefinitely, to suit occasional shut-downs, without danger of the engine giving trouble; and it necessitates shutting down from one to three hours. Where reliable or continuous service is necessary, the required opportunity and leisure for periodical inspection can be obtained only by having a spare unit in each group of gas-engines. It is hardly too much to say that this is the one essential feature in planning a gas-engine plant which constitutes the difference between a plant which will be thoroughly satisfactory in operation and one which will be eminently unsatisfactory.

The rated capacity of a steam-engine being approximately at that point of cut-off where it uses steam most economically, there is always available a considerable reserve capacity, at the expense of economy, by admitting steam for nearly full stroke. The most economical capacity of a gas-engine is its absolute maximum capacity, when at every admission it takes the maximum quantity of mixture, in the best proportions of air and gas, and ignites it at exactly the right point. The adjustments necessary to obtain all these conditions with certainty at every stroke, are too fine for any but test-conditions. To allow for

variations, which are inevitable, even under the best conditions in commercial working, the maximum capacity should be discounted about 15 per cent. to get a figure which may be called "maximum commercial capacity;" and nothing more than this latter figure should be counted upon for those short periods which occur, even with the most uniform load, when the load rises to a maximum greater than the average. In other words, this commercial maximum capacity should equal the peak of the load curve. With a very variable load, therefore, an engine-capacity is required greatly in excess of the average power-demand; and any method of averaging the power-variations of individual loads by driving several such loads from one group of engines will reduce the total engine-capacity to be installed. The central power-house plan with electrical distribution, is, therefore, one which is well worthy of careful consideration for at least some portion of any gas-power plant, in spite of the heavy tax to be paid in transformation- and transmission-losses: losses in the electric generator, the line and the motor, which practically amount to 25 per cent. of the brake horse-power delivered by the engines.

When the original load on a gas-engine which is belted directly to the shafting, outgrows the capacity of the engine, it is generally difficult to devise satisfactory means of supplementing the engine-power. There is a fixed charge of cost and trouble attached to each engine, irrespective of its size, which makes it undesirable to use engines much smaller than the average size. This is frequently a hampering condition. Gas-engines are not self-starting, and cannot be started with the load on. Tight and loose pulleys will answer for small engines, but those of more than moderate power need clutch pulleys on the shafting. They need good foundations, and occupy a floor space large in comparison with their power. Additions generally require floor- or shafting-space which is not conveniently available. And furthermore, when several engines are driving directly on the same shaft, it is difficult to be certain that the proportion of gas to air, and other adjustments, are so regulated that each cylinder is doing its full share of the work. In fact it may be taken as certain that the combined capacity obtained from such a group of engines will not be equal to the sum of their individual capacities. In their adaptability to meet

changing power-demands gas-engines are much less flexible than steam-engines.

The ease with which gas may be transmitted to scattered points without loss, and the heavy power-loss involved in transforming and transmitting the power by electric distribution, at first sight make it appear as though the advantages of using scattered gas-engines, directly connected to the shafting, would far outweigh any benefits obtainable by electric distribution from gas-engines concentrated in a power-house. But when the peculiarities of the engines themselves are considered, in many, not to say most cases, the balance of advantage is found to be on the side of electrical distribution, notwithstanding the cost of electrical apparatus and the constant electrical loss of 25 per cent. of the power delivered by the engines.

Gas-engines need an attendant always within easy reach, and preferably always on the spot; their concentration in a central power-station therefore largely reduces the cost of attendance. The central station gives all the advantages attached to having all engines of the same size. Growth is by additional units. Spare engines for maintenance, the necessity of which has been previously dwelt upon, are minimized; since one spare unit in the central station gives that freedom for repairs, which, with scattered engines, is obtainable only by having a spare unit at every group where reliable service is essential. The variations of individual loads are averaged in the central power-station, so that its load-factor—*i.e.*, the percentage which the average is of the maximum load—is the greatest the character of the whole plant will permit. This reduces to the lowest possible figure the proportion of reserve to total engine capacity, as compared with isolated engines or groups of engines, each of which must contain in itself all the reserve capacity required for the variation of its individual load. And finally, electrical connection is an ideal mode of coupling gas-engines together. It dispenses with all shafting and clutches, while the ampere-meter on each generator gives the engine-man a perfect measure of the work each engine is doing, and so allows him to adjust them for the best individual results.

At points of power-consumption, the mechanical characteristics of electric motors are in striking contrast to those of scattered gas-engines. Motors of any size may be used appro-

priate to the work to be done. They are compact, and need no other foundation than a stiff floor. The motor does its work without attendance, and is easily stopped and started by any workman. It is probably the most robust of all forms of driving-machines, and will stand an extraordinary amount of abuse and neglect without serious damage, and with nearly unimpaired efficiency. It has a large temporary reserve-capacity in excess of its rating, and varies little in efficiency from light load to overload. And if a local increase of power is needed, generally two or three hours' work is all that is required to supplement a motor with a smaller one, or replace it by one of greater capacity.

Each plant and locality presents its own problems, which must be individually considered and suitably met. The question of concentrated *versus* scattered gas-engines admits of no sweeping decision. For those scattered points of a plant where the loads are small in amount or very variable in character, and for points where the loads are liable to considerable change in course of time, electrical distribution is certainly preferable. But if at some point there is a load of uniform character and permanent quantity, and of such amount that the number of engines required justifies a spare unit and gives a reasonably economical cost of attendance, the balance of advantage will almost certainly be in favor of using, at this point of consumption, engines directly belted to the shafting. Between these two extremes will always lie a debatable ground where the decision must rest, not upon calculation, but upon judgment.

## The Gold-Mining Districts of Central Siberia.

BY LEWIS BLANCHARD BROWN, ITAT, SIBERIA.

(New York Meeting, October, 1903.)

### INTRODUCTION.

It is my purpose in the following paper to describe the south-central part of Siberia, with special reference to its geology and the mineral deposits. This region is known politically as the Achinsk and Marinsk districts of the Tomsk Government. It is situated among the Altai mountains, in lat.  $50^{\circ}$  to  $55^{\circ}$  N. and long.  $87^{\circ}$  to  $93^{\circ}$  E., not far from the northern frontier of Mongolia, and is divided as follows:

*Mining Districts.*—(1) The Tomsk mining district, including the political districts of Altai (belonging to the private crown lands of the Russian Emperor) and Marinsk; (2) the Achinsk-Minusinsk mining district, including the political districts of Achinsk and Minusinsk; and (3) the North Yenisei mining district, situated in the northern part of the Yenisei political district of the Yenisei Government.

Fig. 1 is a map of the Tomsk mining provinces, showing the useful mineral-deposits, excepting those of gold, and giving the routes of the railroads and post-roads connecting the various districts.

*Political Divisions.*—The Tomsk Government is sub-divided into seven “oojesdy” or districts, as follows: Barnaul; Biisk; Kuznetsk; Kainsk; Kolyvan; Marinsk; Narim; and the Altai district (private land, as above).

The Yenisei Government is sub-divided into five “oojesdy” or districts, as follows: Krasnojarsk; Kansk; Yenisei; Achinsk; Minusinsk.

Concerning the geology of the region, in addition to the record of my own observations, references will be made to the works of earlier writers. These have been consulted for the purpose of comparison, as regards the districts which I have actually traversed, and for the purpose of completing my account of such portions as I have not been able to visit.

The works of engineers and geologists who have previously visited the Altai, will be freely quoted. The data and figures obtained by my own personal experience in the Altai have been supplemented by those given in the published reports on the industries of the mining districts of the Ural and East Siberian districts.<sup>1</sup>

Apologies are due to the Russian geologists who have pursued investigations in the Altai region, especially Messrs. A. Inostrantseff and A. Zaitseff. Their excellent work is now in progress, and the published results, maps, etc., were not accessible to me in the preparation of this paper.

My experience in the country under consideration is the result of my connection with the gold-mining industry in the Achinsk mining district, and a consequent two years' residence in the Altai.

#### CLIMATIC CONDITIONS.

The region of southern Siberia is characterized by a temperate climate, healthful and invigorating. It is not subject to the great extremes which are experienced in northern Siberia.

Snow covers the bottoms of the valleys for half the year. The average temperature during this time, from October 1 to April 1, is about zero F., and the average for the remaining six months is about 55° F. The working season for mining is from April 1 to October 1. The average depth of snow in winter is 6 ft. The humidity is slight.

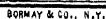
#### TOPOGRAPHY OF THE ALTAI.

The Altai presents itself in the form of a mountainous mass, forming an irregular half-circle, the concavity of which is turned toward the west. Using the name Altai, I refer to that

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<sup>1</sup> Notable among such works of reference are :

*Voyage dans L'Altai Orientale*, P. Tchihatcheff, Paris, 1845. *Asie Centrale*, A. von Humboldt, 1843. *The Gold-Mining Industry of the Tomsk Government*, M. Shostak, 1896. *Guide for All Siberia*, W. A. Dolgorookof, Tomsk, 1900-01. *Collection of Statistics of the Mining-Industry of Russia*, 1897, St. Petersburg, compiled by A. Loranski. *Asiatic Russia*, G. F. Wright, New York, 1902. *The Industries of Russia*, compiled for the World's Columbian Exposition, St. Petersburg, 1893, translator, J. M. Crawford. *L'Or en Sibirie Orientale*, Paris, 1897, E. D. Levat. *Auriferous Gravels of the Sierra Nevada of California*, J. D. Whitney, 1880. *Les Gisements Aurifères de Sibérie*, Baron René De Batz, Paris, 1898. *Les Placers Aurifères de la Sibérie*, A. Bordeaux, *Revue Universelle des Mines, La Métallurgie*, etc., Feb., 1903. *Reports of the Director of the United States Mint*.







part which is bounded on the west by the river Ob, and on the south by the northern water-shed of the Sayanskii mountains. The Abakan and the Yenisei rivers form the eastern boundary, while the line of the Siberian railway encloses it on the north. The Altai, as thus defined, is cut by a large number of rivers which may be classed among the first in the hydrographic systems of the known world. Thus the Ob, which is not the largest river of Siberia, is 2,000 miles long, and the Yenisei, if we take into consideration its longest tributary, the Angara, is about 4,000 miles long.<sup>2</sup> The principal river-systems of the Altai—the Ob and the Yenisei—cover the whole country with a network of streams; the chief tributaries being remarkable both for their extension and the number of their feeders. In the summer all of the rivers are navigable; while in the spring, steamboats travel well up into the mountainous districts, as far as the Chinese frontier.

In general, the topography of the Altai presents the form of old mountain districts. The dominating mountains are rounded, and the valleys, while characterized by great length, have very little grade. Such features as waterfalls, for example, are found only in the high regions of the far south, all the main portion lying to the north of the latitude of Kusnetzk, exhibiting low dome-like mountains and a lack of cañon topography. These domes, which rise from 1,000 to 3,000 ft. above the general level, are intersected with long, wide valleys, in which the present streams tend to lose themselves, and meandering, produce numerous curves and cut-offs. Further north, along the line of the railway and to the north of it, these domes are reduced to mere rolling hills, the country becoming a typical prairie. Mr. Wright has called attention to the base-leveling which exists in various parts of Siberia; and the marks of this, indicated by benches, are frequent in the Altai. Subsequent elevations have occurred, and bench-gravels, similar to those which have been worked for gold in California, are found along the various tributaries of the Yenisei river. Along the stretch of the Yenisei river from Krasnojarsk to Minusinsk, continuous stretches of benches, at a height of 300 ft. above the

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<sup>2</sup> This may be compared with the other large systems of the world, namely, the Nile, 3,370 miles long, the Mississippi and the Missouri together, 4,200, the Amazon, nearly 4,000.

present stream, may be seen for many miles. They are evidences that the country underwent elevation after it was once worn down nearly to base-level. Tchihatcheff, in an illustration of the Katun river,<sup>3</sup> gives an excellent example of high benches along the river, at a considerable height above its level. There is little doubt, as Prof. Wright<sup>4</sup> points out, that serious orogenic disturbances have taken place in Siberia in comparatively recent geologic time.

One striking feature exhibited in the Altai, about the headwaters of the river Tom, is the apparent influence on topography which the weathering of the dioritic rocks appears to exercise. As far as I have seen, many of the great domes, appearing like the tops of half-sunken spheres, are composed entirely of diorite of fine grain. The last remnants of alpine topography are represented by isolated masses of rock, of small dimensions, which appear as castellated masses set on top of the mountains. From a distance these appear as glacial erratics; but, on closer examination, they are seen to be parts of the mountain mass upon which they rest.

Local development of cañon-topography is found in isolated parts of the area which I have visited, but the impression of great age and long continued erosion is so overwhelming that these relics count for little in the general make-up of the country.

Mr. C. W. Purington has called attention to the fact that there are no indications of the former presence of glaciers in western Siberia, except in small local instances.<sup>5</sup> According to all observers who have since mentioned the subject, glaciation is almost absent. The Russian geologists have traced a marginal line in northern Siberia, to the north of which glacial drift occurs. M. Bordeaux quotes the Russian engineer A. P. Guérassimoff, who refers to the evidence of glaciation in the auriferous districts of the Vitim and Olekma rivers (*op. cit.*, p. 148). So far as my observations go, evidence of general glaciation in the Altai is lacking. The southern Altai contains, in high valleys, existing glaciers, but these are of purely local character, like those of the Alps or most of the glaciers of Alaska.

<sup>3</sup> *Op. cit.*, plate No. 1.

<sup>4</sup> *Op. cit.*, vol. ii., p. 505.

<sup>5</sup> *American Geologist*, vol. xxvii., 1901, p. 45.

## PETROGRAPHIC FEATURES.

*Igneous Rocks.*—The granites of the Altai are of three different varieties: first, those rich in amphibole, especially a syenite-granite of the Katun; second, a normal granite of medium grain, with the color generally white or reddish; third, porphyritic granite, especially well developed in the western part. It is very probable that the porphyritic granite was the latest. Gneiss is not common in the Altai, except as a modification of the granites. In the Bazaikha valley, opposite Krasnojarsk, on the east bank of the Yenisei, a granite, colored red by large crystals of feldspar, forms several prominent headlands. Diorite occupies considerably less space than the granites, but its influence as an eruptive agent is greater. The diorite forms the prominent masses in the eastern portion of the Altai. According to Tchihatcheff, it may be said for the Altai in general that the most ancient igneous rocks are allied to granite, while the more recent approach trachyte<sup>6</sup> in composition. Not only does the diorite appear to owe its origin to two great lines of dislocation, which mark the two chains of the Alatau and the Salair, but everything indicates that it is present in great masses, forming a central core for the grauwackes and clay-schists which cover an immense area in the eastern Altai. Tchihatcheff evidently attributes the great rounded forms and immense "intumescences" which form the characteristic type of the mountainous regions of the Altai, to the original laccolitic forms assumed by the cooling masses of diorite.

Besides the normal diorite there are local occurrences of dioritic porphyry, as, for example, in the valley of the Chuya and on the northern banks of the Salair. It appears that these porphyries are intimately allied with the homogeneous diorite, and that the same age must be assigned to the two rocks. The group of porphyries, constituting one of the most interesting portions of the Altai, are of two classes: first, quartz-porphyries; second, melaphyres. The porphyries of the first class are developed in the Zmief country and in the Korgon Alps of the western Altai. In the Zmief district they form

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<sup>6</sup> It is likely that the term *trachyte* as here used refers to a class of rocks which would be differently named under the modern system of petrographic nomenclature.

the country-rock of the metalliferous deposits. Porphyries of the second class, developed in the eastern portion, are of a more basic character, and are frequently amygdaloidal in structure. Typical localities are,—the border of the Jebach river, near the source of the Chuya, between the river Abakan and its tributary the Tachua, and on the two banks of the Yenisei between the villages of Baikalovo and Tesse. These porphyries are always impregnated with calcareous substances and crystals of labradorite and pyroxene are especially well developed.

The melaphyres of the Altai, according to Tchihatcheff, appear to offer an analogy to those of Ararat.

Masses of serpentine, evidently secondary after some of the more basic igneous rocks, may be observed on the right bank of the Chuya. They are intimately mixed with large masses of calcite, and the whole region is colored red by iron oxidation to such an extent as to have received the name of Krasnogora, or Red mountain.

*Quartz.*—A remarkable feature of the Altai is the occurrence of immense masses of quartz. These are connected with schistose rocks, and are sometimes hydro-micaceous and sometimes granular; they are generally of a striking white color. Tchihatcheff refers them to eruptive origin. A remarkable example is to be seen on the banks of the river Chulichman, and near its junction with the Chulcha. Here the quartz forms a hill by itself, and the pure white rock is ribbed with parallel black bands. The most remarkable feature of this quartz is its association with the amphibolite stock. Tchihatcheff believes that volcanic activity has existed up to a recent date. I have seen fresh-looking porphyries in the Altai, resembling andesites, although they have not been microscopically examined. These occurrences are especially prominent about the head-waters of the Oosa river, a branch of the Tom. One variety contained phenocrysts of a light-blue mineral, of amorphous appearance, resembling copper silicate. It is to be regretted that specimens of this porphyry are not at hand at the present time, as they deserve a fuller description.

*Sedimentary Rocks.*—Along the left bank of the river, between Krasnojarsk and Yeniseisk, there are beds of coal, evidently of Cretaceous or Tertiary origin. To the west of Kras-

nojarsk there are red-clay and sand-beds containing concretions of red jasper, probably Cretaceous beds. On the road from the station Itat of the Trans-Siberian railway, leading south into the Achinsk government, one passes an area of red sandstone beds which are presumably Triassic. These beds are seen always dipping to the east, and are abruptly broken off on their west side. The similarity between this topographic feature and the fact mentioned by Tchihatcheff, that the right banks of the rivers are always the higher in Siberia, may be mentioned in passing.

In traveling south from the railway station of Itat, after traversing a wide valley of steppe-character, containing many lakes, where the red-sandstone beds are continually exposed for nearly 100 miles, one comes to the foot-hills of the Altai proper. As one approaches the Altai, to the west, the red sandstones, dipping continually to the east, are underlain by the limestones, which form prominent outcrops in all the eastern foot-hills. These limestones are referred by Tchihatcheff to the Carboniferous and Devonian. In the valley of Bazai-kha, on the east bank of the Yenisei, opposite Krasnojarsk, highly tilted beds of sandstone and limestone occur, having a strike N.  $60^{\circ}$  E. and a dip  $30^{\circ}$  SE. One variety of the sediments is highly bituminous (*stinkstone*). On the west side of the valley innumerable veins of calcite penetrate these sediments, and there are copper stains along the cliffs, in connection with beds of highly crystalline limestone.

Rocks of a metamorphic character are widely developed in the western Altai, but not so much as in the central and eastern parts. These metamorphic rocks are composed mostly of clay-schists, quartz-schists and metamorphic limestones with their various modifications.

Limestones have a great development in the eastern portion of the Altai, and are presumably of Carboniferous age. Although not greatly tilted, they are always more or less metamorphosed in character. It is not likely that localities from which fossil remains may be obtained are plentiful. Tchihatcheff refers to the limestone-beds as beautifully exposed in cliffs of Carboniferous and Devonian ages, along the Yenisei to the north of Minusinsk. He calls attention also to the fact that the Carboniferous limestones in the central and western

Altai contain metalliferous deposits. These will be referred to later. Organic remains are common in the alluvial deposits of the Altai, and numerous bones of elephants and rhinoceros are found at widely separated points. The auriferous gravels of the Alatau, the vast deposits which surround the town of Barnaul and those of the river Alay are notable in this respect.

*Weathering of the Rocks.*—Some attention should be called to the remarkable weathering observed in the Altai. Forces of disintegration may be expected to play an important part in a region subject to great extremes of temperature, and in receipt of a large amount of precipitation. The weathering of the large masses of granite and diorite, previously referred to, may be seen to great advantage, as there is no other agency than that of the slow-running streams to carry away the detritus. One remarkable effect of weathering is apparent in the immense boulders which rest on the tops of the mountains. These have already been mentioned. Another remarkable result of the same process was seen near the Petrovka creek, a tributary of the Mali Oos. Here, what appears to be a long ridge 75 ft. in height, and half a mile in length, extending out from a high rock-mass, was taken for a glacial moraine or kame. A section from which the soil had fallen away looked like a bank of sand. Upon investigation with a pick it proved to be merely arkose resulting from a granular rock. It crumbled between the fingers like sand.

#### MINERAL-DEPOSITS AND THE MINING INDUSTRY.

The mineral-deposits of Siberia in general have been so fully described by different writers, that I shall only refer briefly to those which are outside of the district and which I have personally examined. As is well known, gold, platinum, silver, copper, iron, lead, graphite, tin, and more or less coal of varying quality, occur in Siberia. The deposits have been mined by the Russians with a small show of activity for over 150 years; and although many of them, especially those of iron, have been shown to be extensive and of unexcelled quality, the progress of the mining industry has been slight.

According to the figures prepared by Mr. A. Bordeaux,<sup>7</sup> Russia and Siberia together have produced, since 1745,

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<sup>7</sup> See list of works cited on p. 778.

120,000 poods (1,983,200 kilograms) of gold—presumably up to the end of 1899, which is the date of the latest statistics given in his pamphlet.

The occurrence of gold is wide-spread; but in general only the alluvial deposits have been extensively studied. So far, quartz-veins carrying gold have been found only within limited areas. The quantity of gold extracted from quartz-veins has been small compared to that extracted from placers. Curiously enough, it was from a quartz-vein that the first find of gold was made in Russia, in a portion of that country which was then referred to as Siberia. Gold was first discovered by the Russians, according to Crawford,<sup>8</sup> in 1744, near the town of Ekaterinburg, 6 miles to the northwest, at a village now known as Berezovsk. From this and other descriptions of the find, it appears that this was free-gold in quartz, probably from one of the numerous stringers which penetrate the granite of that region, which granite is generally referred to as beresite. This name, according to Messrs. Nitze and Purington<sup>9</sup> is without special significance. Although the first find was gold-quartz, it was from placers that the first production was obtained. In 1814 alluvial gold was worked in the Urals, for the first time, and in 1820 in the Altai region of Siberia.

Subsequently the eastern part of Siberia underwent gradual development. The Yenisei districts, the region of the Amur, the coast-provinces, and the Lena River country have for fifty years been the most productive portions of the Siberian gold-fields. The Altai gold-fields will presently be described, but a few words should be said with reference to the silver-lead deposits of the western Altai, which in the early part of the 19th century were of some importance.

*Altai Silver-Deposits.*—The silver-mines of the central Altai are situated for the most part in a Carboniferous limestone. They occupy veins traversing beds at various angles, the veins having a strike usually northwest. They carry silver-bearing lead-ores mixed with a large proportion of barite gangue. Silver tellurides are also mentioned as occurring in these veins. These mines have been worked out to depths of 700 ft., and

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<sup>8</sup> Crawford, *op. cit.*, vol. iv., p. 11.

<sup>9</sup> The Kotchkar Gold-Mines, *Trans.*, xxviii., 26 (1898).



it is to be inferred from the general cessation of operations that the ore-deposits did not pay in depth.

The silver-mines of the Salair mountains contain the values in galena associated with barite. At the present price of silver,<sup>10</sup> the average tenor would be only \$3.50 per ton. There are silver-mines in the Kolyban district similar in mineral character, but they are much richer. The Zmeeff (Zmief?) mine, near Kolyban lake, south of Barnaoul, contained a shoot of ore 400 ft. long and 150 ft. wide which was worked to a depth of 400 ft. The average tenor of the ore is not given, but at the present price of silver, portions of it ran as high as \$400 per ton; or, in Russian figures, one pound,<sup>11</sup> 64 zolotniks, to the 36 lb. of ore. Another rich group of mines is the Rydirsk group; in one of which, the Sokolnoi mine, the barite was replaced by hornstone at a moderate depth. This hornstone is said to contain free gold to the extent of \$3 per ton when tested by simple crushing and assaying. The deepest workings mentioned by Tchihatcheff are a little less than 700 ft.

In 1845 the production of silver in the Altai was about 50 poods, equivalent in value at the present price to \$12,500, so at that period there was apparently but little mining activity.

#### GEOLOGICAL CHARACTER OF THE ALTAI GOLD-DEPOSITS.

*Veins.*—Little can be said regarding the primary deposits of gold in the Altai region, for as yet they have been exploited only to a small extent. Of those deposits which I have seen, the most remarkable was at the head of the Fyedorovski creek, a tributary of the Chulim river. The veins recently found here have no definite walls, but appear to consist of irregular masses of quartz, from 1 to 6 ft. wide. The quartz contains a little pyrite, is white and glassy in appearance and contains free gold to an unusually large extent. Several veins of this character have been found at the head of the creek.

There is little doubt that the basic porphyries of the eastern Altai have been more or less directly connected, as regards origin, with the gold-bearing veins which have recently been found

<sup>10</sup> Assuming this to be \$0.50 per troy ounce.

<sup>11</sup> A "funt" equals 0.9028 lb. avoirdupois.

there. In the instance noted, at the head of the Fyedorovski creek, a tributary of Chelim river, native gold was found in diorite, by Professor Zaitzeff of the University of Tomsk, in the summer of 1900. I examined this occurrence a few weeks later, and can vouch for its authenticity. No vein of quartz was to be seen within 500 ft. of the occurrence, and the gold was apparently bedded in original particles in the diorite itself. There was, however, in the immediate vicinity, an immense mass, either a dike or a stock, of an exceedingly basic rock, evidently a peridotite having a somewhat porphyritic structure. At a distance of less than 1,000 ft. remarkable reefs of gold-bearing quartz have been uncovered which have been notably productive. Such an unusually well-defined association of gold-bearing veins with igneous rocks, from which the former seem to have drawn their metallic content, is worthy of careful attention.

There is no case on record, so far as I know, of what appears to be so clearly defined a connection between gold and its immediate source. Tchihatchef calls attention to the theory that the presence of diorite is always favorable to the occurrence of gold, and gives numerous examples drawn not only from the Altai but from other parts of the world, to support his hypothesis. It is possible that the lateral-secretion theory has here a striking and immediate application.<sup>12</sup>

Near the head of the Sorela Oos river, quartz-veins from 3 to 15 ft. wide have been found, many carrying free gold. They have regular walls, and occur in a country-rock of diorite. The data regarding these and other quartz-veins carrying gold are necessarily meager, as the outcrops are, in most cases, the only portions visible. A further description of the Sorela valley will be given later in this paper under the head of "Individual Placers."

In general, it is probable that in the Altai, as in other portions of Siberia, the gold found in the placers does not come from large veins so much as from innumerable small stringers of quartz in the vicinity. In the course of a journey through

<sup>12</sup> Compare the occurrence of gold in granite noted by W. P. Blake, *California Minerals*, 1864; G. P. Merrill, *American Journal of Science*, iv., p. 309, 1896; J. B. Jaquet, "Geology of the Broken Hill Lode," *Memoirs Geological Survey of New South Wales*, No. 5, Australia, 1894; Möricke, *Tschermak's Mineralogische und Petrographische Mittheilungen*, iii., 1891, p. 186.

the old placer-districts of the Crown Canals in the Altai, almost no traces of veins were seen which could account for the source of the alluvial gold, although thousands of acres of bare diorite and granite were visible on the tops and sides of the mountains. Mr. Guérassimoff, quoted by Bordeaux (*op. cit.*), says, that in the Olekma and Vitim districts, the auriferous gravels contain much fine pyrite which assays high in gold. This pyrite occurs as films and knife-blade seams in the underlying schists, without visible accompanying quartz. The large quartz-reefs of these northern districts are notably barren of gold-values.

*Placers.*—These are the result, like all the Siberian auriferous gravels, of an immense amount of erosion, which has been interrupted by periods of elevation. Compared with the Ural gravels, those of the Altai are very much thicker, and occur in wider and longer valleys. As shown by Tables I. and VII., they are exceeded in thickness only by those of eastern Siberia.

The gold alluvium of eastern Siberia is covered likewise by a great thickness of the turf, so called, which is described below in connection with the Altai placers. This turf, which is sometimes true peat, probably attains a thickness of 50 or 60 ft. in the Lena regions, although true information of its actual thickness is hard to obtain, on account of the loose nomenclature employed by the miners. An additional reason is that the turf and overburden are not stripped in placer-operations, as is done in the Ural and western Siberia, but the gravel underneath is mined by means of shafts which are sunk through the upper barren alluvium.

The placers of the mining-districts covered by this paper may be referred to three water-systems. The first is that of the area drained by the Kiya, which lies in the Marinsk mining-district. The Kiya is a tributary of the Chulim, which flows north and west into the Tom, a tributary of the great river Ob, one of the three chief rivers of Siberia, emptying into the Arctic Ocean. The second and third are the areas drained by the Byeli, or White Oos, and the Chorni, or Black Oos, also tributaries of the Chulim, but flowing in farther to the east. This country, comprising the second and third groups, is in the Achinsk mining-district.

As has been noted in the general description of the Altai

topography, the mountains in the region drained by the Kiya, in the first district mentioned, are less rugged than the mountains of the second and third districts, lying in the Achinsk district. This difference may be accounted for by the fact that the mountains lying in the Kiya district are farther away from the great central elevated core of the Altai, while those of the Achinsk placer-district are nearer. In comparison with the elevation, there has been no more erosion in one district than in the other, but the original elevation of the one group was less than that of the other.

An interesting corollary of this fact appears in Table I., given below, which indicates that the gravels of the Marinsk mining district are much thinner than those of the Chorni Oos district. It thus seems that in the Achinsk or eastern district, the amount of residual material represented by the gravels filling the valleys, has been greater. In the Marinsk district, on the other hand, the low mountains are accompanied by a smaller amount of detritus. It is probable that, as a whole, the eroding agencies were at work on a more active scale in the eastern than in the western and lower region,—an effect which would be explained by the greater height of the mountains and the consequent more severe climatic conditions.

Table I., translated and reduced from Loranski,<sup>18</sup> shows the depths of the stripping and gold-bearing gravel in the three river-systems:

TABLE I.—*Statistics of Siberian Gold-Bearing Gravels.*

River-System.	No. of Cubic Yards Worked.	No. of Mines.	Workmen.	Total Yield.	Yield Per Cu. Yd.	Average No. Workmen per Mine.	Stripping.	Pay Dirt.	Average Value per Cu. Yd. from Surface to Bed-rock.
Kiya.....	311,000	75	835	\$229,000	\$0.70	11	Ft. 9	Ft. 6	\$0.28
Chorni Oos.	167,000	24	427	76,000	0.40	17	12	8	0.16
Byeli Oos....	87,000	13	353	47,000	0.54	27	8	5	0.20

SECRETARY'S NOTE.—The volume and yield in this table are apparently given in round numbers, as the result of approximate reduction to yards and dollars.—R. W. R.

It will be seen from Table I. that the depth both of the gravels and the stripping is less in the Kiya than in the Chorni

<sup>18</sup> *The Collection of Statistics of the Mining-Industry for the Year 1896*, A. Loranski, St. Petersburg, 1899.

Oos. As regards the Byeli Oos, which is in the same district as the Chorni Oos, the shallow depth of the material is due to the fact that the workings followed the small tributary gulches of the streams. The operators were unable to mine in portions of the streams which corresponded to those worked in the Marinsk district. This is due to the fact that the lower gravels of the Byeli Oos tributaries are extraordinarily deep. Thus the figures of Table I., which appear to militate against this hypothesis, are in reality a strong confirmation of it. For example, the Semni placer, in the upper part of the Zhelezni creek (one of the tributaries of the Byeli Oos), showed 6 ft. stripping and 6 ft. pay-gravel, while the Blagodatny (a gulch of the Tek Tirik, another tributary of the Byeli Oos, higher up from the mouth) showed only 2 ft. stripping, and 4 ft. gravel. At the mouth of the Blagodatny gulch on the Tek Tirik several pits were sunk during the winter months, to the depth of 70 ft. without reaching bed-rock, although from the richness of the gravel at the bottom of the pit it was evident that the bed-rock was not far away. On the Zhelezni creek, where workings are now carried on, the depth to bed-rock is 46 ft. From these and other examples which might be quoted, it is evident that deep gravels in the Byeli Oos system are common.

On the Kiya, operations are conducted in a small way, as may be seen from Table I., in which the number of men employed is given. The district is peopled by Jews, who are generally small operators in Siberia. In the Achinsk district, the placers are all in the hands of people with more capital, and the work is conducted generally on a larger scale.

*Production.*—With regard to the yield, since the preparation of Table I. in 1896, extensive operations on the Byeli Oos system have increased the output to approximately \$600,000 for two mines alone.

#### CHARACTER OF THE AURIFEROUS GRAVEL-DEPOSITS IN GENERAL.

As is well known, the alluvial deposits of Siberia in general are characterized by a covering of matted earth and vegetation, commonly referred to as "turf." For the most part this turf is the material known in America as peat, which may be dried and used as fuel. Among the gold-miners, however, the distinction between true turf and other material is not sharply

drawn. All material in the placers which overlies the "plast" or pay-gravel is included under the name "tourf," which may be poor gravel, rounded or angular in character, and occasionally sand; or even landslide material.

The turf, properly speaking, varies in thickness from 6 in. to 60 ft. Data from the Lena district in eastern Siberia give an occasional depth of 150 ft., but I am not sure that this is true turf. It is probably, in part, barren gravel. Analogies between the eastern Siberian deposits and those of Alaska, especially those of the region behind Cape Nome, appear to exist to a remarkable extent.<sup>14</sup> The "tourf" of southern Siberia is not analogous to the tundra of Cape Nome, which is described as moss. However, it is possible that the moss occurring in the high basins of the Altai is of the tundra type.

The turf is rarely absent even on the barren and treeless steppe-area about Troitsk, in the Orenburg Government, where it exists in the depressions of the plain, as a thin covering. The turf occurs not only in the valleys of the country, but is also on the very summits of some of the Altai peaks, which exceed 7,000 ft. in height. It exists also on the sides of the mountains, and holds water like a great sponge. To this fact may be attributed the remarkable scarcity of visible streams and small brooks. A journey of a whole day along the mountain ridges and through the valleys of the Altai will hardly furnish opportunity for seeing more than a dozen brooks tributary to the large creeks.

In the gold-bearing valleys of the Marinsk and Achinsk districts, the true turf does not exceed 2 ft. in thickness. Underneath the turf is a layer of rounded gravel, consisting of débris brought from a distance, which is occasionally mixed with sand and layers of tough clay. Beneath this is a layer of gravel of varying thickness, which is generally barren. Underneath this gravel is the pay-gravel, from 2 to 8 ft. in thickness, usually of an angular or sub-angular character. It resembles the ordinary auriferous gravels of Siberia, but in comparison with those of California, it appears to have suffered much less wear and to have traveled a relatively much shorter distance.

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<sup>14</sup> Frank C. Shrader and A. H. Brooks, "Preliminary Report on the Cape Nome Gold Region, Alaska," *U. S. Geological Survey*, 1900, pp. 14, 43.

A feature which characterizes the Siberian deposits is their remarkable similarity over wide stretches of country. Thus, from Levat's descriptions, it appears that the gravels of the Amur district are in a few places almost exactly similar to those which I have seen in the Ural and in western Siberia, the only difference being in the tenor of the gold and in the fact that the eastern Siberian gravels are generally frozen throughout the year. In this regard see also DeBatz's<sup>15</sup> descriptions of the Amgoune river-deposits.

#### METHODS OF MINING.

Tchihatcheff saw ancient workings in the Altai, and attributes them to a people known under the general name of "Chudi." He gives no description of these workings, however. After this pre-historic work done in the Altai, there is no mention of precious metals being worked there until the second part of the seventeenth century. Veins of silver-ore were first discovered in this district by Demidoff in 1735, and mines were worked continuously after this for more than 100 years. Some silver-mining is still carried on, and more silver is produced from the Altai district to-day than from any other portion of the empire, the amount being about three-quarters of the entire product. All the silver is obtained from the argentiferous lead-ores previously described.

In the Altai, deposits of lead have been worked only in connection with the production of silver. Copper-mining, and smelting, was begun by Demidoff at the beginning of the eighteenth century. The production of copper from this region has never been as great as that from the Ural, although the mines of the western Altai have been worked for more than a century. The annual yield from the whole empire to-day does not exceed in value \$1,500,000, or 10,836,000 Russian pounds. Of this, 720,000 Russian pounds came from the Altai in 1890. This proportion, 7 per cent., remains about the same, but the product is increasing every year. As is the case with the iron-mines in the Ural, the copper-deposits of the Altai and adjacent country are too far from the present routes of transportation to be successfully worked on a large scale.

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<sup>15</sup> De Batz, *op. cit.*, p. 104.

Zinc has never been worked in Siberia, although unimportant discoveries have been made.

Quicksilver has never been found in any quantity, although traces of it exist in many parts, as, for instance, near Minusinsk, also in the Nerchinsk district and along the shores of the Okhotsk sea. No deposits are worked, the Russian quicksilver coming from the Auerbach mines, of the Ekaterinoslaf government in southern Russia. It is a curious fact that Siberia has got a portion of its unfavorable reputation from the fact that convicts are worked in quicksilver-mines, "the mines of mercury," as Kipling<sup>16</sup> calls them. Where these mines are located, or where their product goes to, the mining statistics do not show.

Iron-ore is very little worked in the Altai, and no such extensive deposits as exist in the Ural have been found there. On the Abakan, iron-ore of a very fine quality is mined and smelted, but the yield of the furnaces is unimportant. The iron, according to analysis, contains practically no phosphorus. There are a few other small deposits, but the entire production is so unimportant that it is not worth comparing with the iron-industries of the Urals and of southern Russia. Only about two-thirds of 1 per cent. of the Russian iron-product comes from Siberia.

Graphite has been worked for many years in Siberia, but the production is limited by the demand. The deposits are of good quality and of unlimited extent.

The earliest discovery of gold in the Russian Empire has been referred to, but it was not until the beginning of the nineteenth century that gold was worked in Siberia. It was found in 1829 on the eastern slope of the Alatau mountains, a portion of the Altai. This date refers, of course, to the discovery by the Russians, because, as I have already stated, it is probable that gold was mined by the ancients.

Private mining was allowed in the Urals in 1819; and in 1838, Siberia was thrown open to private gold-washing, with the exception of certain lands in the Altai and Nerchinsk districts, which were reserved for the private treasury of the Crown.

The following statistical data have been partly taken from the volume of the *Industries of Russia*, J. M. Crawford, 1893.

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<sup>16</sup> The Rhyme of the Three Sealers, *The Seven Seas*, Rudyard Kipling, 1899, p. 60.



TABLE II.—*Siberian and Ural Gold-Product from 1816 to 1890, by Intervals of Five Years.* (Crawford, iv., p. 13.)

Years.	Production of Gold.	Average Yearly Production.	Years.	Production of Gold.	Average Yearly Production.
	Value.	Value.		Value.	Value.
1816-1820	\$760,000	\$150,000	1856-1860	\$73,061,000	\$14,612,200
1821-1825	6,070,000	1,214,000	1861-1865	73,500,000	14,700,000
1826-1830	13,284,000	2,654,800	1866-1870	92,110,000	18,422,000
1831-1835	18,288,000	3,657,600	1871-1875	107,580,000	21,516,000
1836-1840	20,255,000	4,051,000	1876-1880	124,010,000	24,802,000
1841-1845	48,456,000	9,691,200	1881-1885	108,270,000	21,654,000
1846-1850	72,846,000	14,569,200	1886-1890	109,940,000	21,988,000
1851-1855	67,940,000	13,588,000			

*From Russian Official Statistics.*

1891-1895,	. . . . .	186,504 kilograms.
(a) 1896,	. . . . .	32,565 "
(b) 1897,	. . . . .	34,977 "
1898,	. . . . .	38,314 "
1899,	. . . . .	33,354 "
1900,	. . . . .	30,312 "
1901,	. . . . .	34,883 "
(c) 1902,	. . . . .	33,905.2 "

(a) The Russian figure, from the *Collection of Official Statistics*, is 1990 poods, 19 funts, 50 zolotniks, 29 doli. For this year (1896) the Report of the Director of the Mint gives a lower figure, namely, 82,404 kilograms.

(b) Reports of the Director of the Mint.

(c) By courtesy of the Director of the Mint.

TABLE III.—*The Comparative Proportion of Gold Produced in the Urals; Eastern Siberia; and in Western Siberia.*

Years.	Urals.	Western Siberia.	Eastern Siberia.
	Per Cent.	Per Cent.	Per Cent.
1861-1865	21.3	4.6	74.1
1866-1870	21.7	6.1	72.2
1871-1875	17.2	7.2	75.5
1876-1880	20.0	6.0	74.0
1881-1885	22.6	6.1	71.2
1886-1890	22.7	7.0	64.2

In Mr. Bordeaux's report it is stated that during the 10 years following 1839, the Yenisei district produced annually 1,000 poods (16,360 kilograms) of gold yearly, chiefly from the Uderei and Mamona Creeks.

Mr. M. Shostak, the former Director of the Mining Department of Tomsk, has compiled some very interesting figures

relating to the mining-district of the Tomsk government. He has calculated that the total amount of gold extracted from the date of the earliest Russian operations in this district up to the date of his report, 1894, reached the vast sum of \$318,000,000. It is probable that over 90 per cent. of this product was alluvial gold.

The greatest yield in any year was \$11,250,000, obtained in 1847. Since then the production has steadily decreased, with the exception of occasional years, such as the last one included in his report. The lowest point was reached in 1889, when the yield was \$3,200,000. The average since 1880 has been about \$3,500,000. The final year of the report gives \$3,798,000.

From Table III. may be seen that the rich placers of the Tomsk mining-district, which includes the principal deposits of western Siberia, were exhausted before 1861.

Table IV. shows the total amount of work done during 1894 in the Tomsk government.

TABLE IV.—*Work in the Tomsk Government During 1894.*

(From M. Shostak.)

Number of Cubic Yards Worked.	No. of Mines.	Mines Unworked.	No. of Workmen.	Total Yield.	Average Yield per Cu. Yd.	Stripping.	Pay Dirt.
4,600,000	670	1,639	22,048	\$3,798,000	\$0.82	6 in. to 60 ft.	6 in. to 40 ft.

The large figures of output in Table IV., and the greater depth of turf and gravel, as compared with Table I., which refers to the Marinsk and Achinsk districts, are due to the fact that the North Yenisei district is here included, which, although belonging politically to the Tomsk district, is entirely different, from a geological standpoint. It lies in an entirely separate area, 400 miles north of the Altai district proper. These figures of Mr. Shostak are interesting, because they show the gradual diminution of the placer-product, as the rich placers worked in the early days became exhausted. No quartz-mining was attempted, and as fast as the rich gulches were worked out, poorer ones were worked in turn, until the economic limit under Russian methods was reached. On the other hand, of

late years, and especially since 1894, the beginning of quartz-mining, and the working of large areas of low-grade gravels by dredging and other methods, is causing a steady increase of the gold-production from this district. For instance, in 1896, when quartz-mining was beginning to become important, but dredging had hardly started, the yield was \$4,086,000.

#### MINING CONDITIONS IN THE ALTAI.

*Working Season.*—Levat<sup>17</sup> gives the working season in the Amur, and eastern Siberia in general, from May 1st until the end of August (Russian style, which is thirteen days behind our calendar). In the southern part of the Amur district and in the southern Trans-Baikal region, the work can be prolonged until well into September. For example, the rivers of Amur at Blagovestchensk are frozen from about the 25th of October to the 25th of April, Russian style. There is but little snow in the Amur district, but the rains of June cause the rivers to rise to a height of 35 ft. or more.

According to my experience in the Altai, the longest working season for placer-operations lasts for six months, from April 15th (Russian style) until October 15th. Snowfall is great in the winter, and there are two periods of flood, one following the melting of the river ice, in April, and the other, in May, being caused by the melting of the snow in the mountains.

In the Urals the season is no longer than in the Altai, and depends largely on the latitude, although the excessively cold weather of eastern Siberia is not found.

In eastern Siberia the methods of working gold-placers have to be adapted to treat the frozen gravel, which exists throughout the year. In those places where the overburden has any appreciable thickness, it is not removed, the gravel being worked by drifting.

*Labor.*—The laborers employed in the Siberian placers, form a class by themselves, and have been for many years accustomed to certain methods of work. This result is due to the conservative methods of Russian operators, who have pursued precisely the same methods since the beginning of the gold-mining industry in Siberia. When new methods are introduced, the men

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<sup>17</sup> Levat, *op. cit.*, vol. ii.

must be handled with great care, owing to their general obtuseness to innovations.

Levat, in speaking of the Siberian operators themselves, calls attention to the fact that the cause of the tardy development of the gold-resources is the disinclination of local capitalists to advance large sums for an extended period. They are ready to lend small sums to good placer-miners, when it is assured that the returns will be had the same year. This spirit of inertia prevents the laborer from being an adventurer. There are few prospectors among the laboring class, and the number capable of any advancement is small.

The Russian government, realizing the proclivity of the men toward drunkenness, makes, for the benefit of the employer, a law requiring a man hired on contract to work continuously for a period of six months or a year, and he is not paid until the end of that period. I have found this law to be beneficial, and, without it, it would be impossible to get any continuous work done by the Russian laborer. At least, this is the case in Siberia.

I consider the Mohammedan Tartars from the Kazan government of eastern Russia the best mining-labor yet employed in Siberia. They are temperate, silent and hard-working. They observe few holidays, and are much more desirable in every way than the Russians. Mr. W. A. Dolgorookof<sup>18</sup> mentions the system for engaging men in eastern Siberia, and as a rule, it appears that they offer their services. Some of the large companies, at distant localities and in less populated districts, engage men at Blagovestchensk and other towns of size, through an agent who receives in advance as a guarantee from 30 to 70 roubles, to cover the traveling expenses of the men to and from the mine. The law seems to be general in eastern Siberia that the mining-companies shall pay the men's transportation to and from the mines. In the Altai, on the other hand, the custom is for the men to pay their own fare.

In eastern Siberia the wages are higher than in the Altai, and besides that, the men are fed by the mining-companies, while in the Altai and Ural they feed themselves.

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<sup>18</sup> *Guide for All Siberia*, p. 276, et seq.

The comparison of the scale of wages and the price for food will be seen in the Tables V. and VI.

TABLE V.—*Wages Paid to Men in Ural, in Western Siberia, and in Eastern Siberia.*

	Ural.	Western Siberia.	Eastern Siberia.
Foremen.....	50 cents and provisions.	60 cents and provisions.	\$1.50 and provisions.
Workmen, first grade....	40 cents without provisions.	50 cents without provisions.	\$1 and provisions.
Workmen, second grade..	25 cents without provisions.	30 cents without provisions.	75 cents and provisions.

TABLE VI.—*Prices for Food and General Supplies, in Ural, in Western Siberia, and in Eastern Siberia.*

	Ural.	Western Siberia.	Eastern Siberia.
Meat.....	3 cents @ pound.	2.5 cents @ pound.	8 cents @ pound.
Rye flour.....	0.5 cents @ pound.	0.75 cents @ pound.	4 cents @ pound.
Wheat flour.....	2.5 cents @ pound.	3 cents @ pound.	6 cents @ pound.
Sugar.....	8 cents @ pound.	9 cents @ pound.	15 cents @ pound.
Butter.....	10 cents @ pound.	14 cents @ pound.	29 cents @ pound.
Wood.....	\$1.05 @ cord.	\$1.10 @ cord.	\$1.40 @ cord.
Charcoal.....	1.35 @ cord.	1.35 @ cord, not delivered.	1.75 @ cord.
Dynamite *.....	70 cents @ pound.	75 cents @ pound.	.....
Oats.....	0.5 cents @ pound.	0.75 cents @ pound.	4 cents @ pound.
Hay.....	\$6 @ ton.	\$5 @ ton.	.....

\* The high price of dynamite is partly owing to the limited demand for it, under the present conditions of the mining-industry.

#### TENOR OF THE GOLD-BEARING GRAVELS.

Table VII., compiled from the work mentioned in the footnote,<sup>19</sup> is based on the records which are kept from day to day by the placer-miners, and turned in from time to time to the Mining Department by order of the Russian government. It is thought that the statistics so obtained are in the main correct; the tables, as a rule, falling short, rather than exceeding the actual product.

The figures here are averaged from a great number of districts, and represent, as nearly as possible, the yield of the placers, the thickness of the stripping and gravel, etc., for the year 1896.

<sup>19</sup> *Collection of Statistics of the Mining-Industry, for the Year 1896, by A. Loranski.*

TABLE VII.—*Production of Placer-Gold for the Year 1896.*

	Number of Cubic Yards Worked.	Mines Worked.	Workmen.	Total Yield.	Stripping.	Pay Dirt.	Average Per Cu. Yd.
Urals.....	3,724,000	291	26,111	\$3,565,040	9 ft.	3 ft.	\$0.96
W. Siberia...	4,600,000	670	22,048	3,654,300	6 in. to 60 ft.	6 in. to 40 ft.	0.92
E. Siberia ...	7,255,000	800	30,965	13,635,000	6 in. to 60 ft.	6 in. to 42 ft.	1.75

TABLE VIII.—*Detailed Yield of Gold (in Kilograms) for the Years 1897, 1898 and 1899 (according to the figures of Mr. Bordeaux), for Eastern Siberia and for Western Siberia.*

	1897.	1898.	1899.
Eastern Siberia: including Birusinsk, Transbaikal, Lena, Amur, and Primorsk mining-districts, and including the Crown domain of Nerchinsk.....	20,076.17	21,153.79	20,304.17
Western Siberia: including Semipalatinsk, Tomsk, comprising Altai, Achinsk-Minusinsk, Marinsk, North and South Yenisei, and Kansk.....	5,216.66	4,967.69	5,478.63

The only restriction placed by the Russian government on the private production of gold is the provision that the gold must not be taken out of the country in its rough state. If the gold-miner sells his gold to the government laboratory in Tomsk, or Irkutsk, he receives full value for it, both for the gold and the included silver. The payment is made at once in the form of a draft on the government bank, which may be cashed in standard Russian gold coin, if desired. The tax of 3 per cent. which formerly existed on gold mined in western Siberia has now been removed, and only nominal charges are made for transportation and coinage.

The operator must transport his gold to the railway at his own risk, but from the railway station it may be transported at the government risk.

The description of the Zhelezni placer-mine, and the operations now conducted thereon, is deferred until a later paper. The Blagodatny placer and Sorela Oos placers are here briefly described.

*Blagodatny.*—This property is located in a side-gulch of the Tek Tirik creek, a branch of the Byeli Oos, a tributary of the Chulim which flows into the Ob.

It lies on the foothills of the Altai, and the country is characterized by hills, which rise to the height of 1,000 ft. above the stream-bed. The region is well timbered with yellow pine, spruce, birch and larch. It is also well watered by the Tek Tirik creek which furnishes 1,500 miner's inches in the dry season. From this minimum the water varies to 3,000 miner's inches in the wet season. At no time does the Blagodatny exceed 50 in. in depth, and its grade does not exceed 3 per cent. The bed-rock, as found in pits and as indicated by outcrops on the hillsides, is, in part, diorite of a prevailing light color. What form this diorite assumes is uncertain, but it is likely that it occurs as stocks of small dimensions and dikes in connection with, and cutting, beds of mica-schist which also occur in the placer. Quartz-layers are included in the schist. The country-rock has been extensively disintegrated, the only fresh occurrence being occasional hummocks on the diorite-floor of the placer, which probably represent original granitic modifications of the rock.

The indication of mineral-bearing veins in the vicinity is slight. There are a few wide quartz-veins, of a singularly barren appearance. It is probable that the alluvial gold now found in this and neighboring placers comes from a great number of minute quartz-stringers, forming a net-work in certain parts of the rock, which generally show the presence of pyrite. The gravel is less than 6 ft. thick, and is in the proportion of 2 ft. of stripping and 4 ft. of pay-dirt. The area worked covers about seven acres.

The output of the placer when worked by Russian methods, which I will presently describe, was about \$100,000. The gravel is angular, indicating very little transportation, and consists of diorite, granite and mica-schist. The gold is 920 fine. According to the Russian figures, the pay-dirt ran one zolotnik to the 100 poods, or \$2 to the cubic yard, which corresponds to about \$1.50 per cubic yard from the grass-roots down.

The property was worked by a Russian, who could neither read nor write; and from his earnings he has amassed a fortune in Tomsk real estate. Last year he staked out a quartz-

claim; believing that the iron-stained mica in his bed-rock was gold. He had some of this assayed at the government laboratory in Tomsk, and it was found that gold was present; to what extent, I do not know. It is not likely, however, that its tenor is sufficient to permit of its being successfully worked.

It will be interesting, in this regard, to describe the Russian practice, even at the risk of repeating what may have been already described in our *Transactions*. One method much in vogue is by means of the *botchkka*, a revolving sheet-iron cylinder punched with 1-in. holes, in which the major part of the gravel is washed. This cylinder is set up on a staging, above which to the height of about 20 ft. is built a bridge with an inclined driveway leading to the ground. The loaded carts or *taraitakas*, each containing about 0.25 cu. yd. of gravel, are drawn from the gravel-pit up this driveway by a horse. The carts are loaded by hand, and dumped also by hand into the hopper at the end of the bridge, which connects directly with the *botchkka*, while the over-size material falls from the lower end of the *botchkka* into a large bin, from which it is carted away. The actual saving of the gold on the sluice-tables is done much as in American placer-mining.<sup>20</sup> M. Levat speaks of the Siberian tables in favorable terms. From the time when the gravel is dug from the bank with pick and shovel, hauled up the long incline in carts, and dumped into the hopper, no mechanical device is used. The result is that the Siberian placer-miner spends a large share of his-working cost for the hand-labor used in his operations.<sup>21</sup>

By the Russian methods, the stripping of the turf as a rule is done in the winter, when labor is cheap. It seems to be the consensus of opinion of all engineers who have given attention to the Siberian methods, that, notwithstanding the low price of labor, the methods of handling gravel are generally to be condemned. Levat quotes the Siberian operator as being of the opinion that the *botchkka* is the most serviceable gold-washing machine that it is possible to imagine.

*Sorela Oos.*—The extraordinary valley of Sorela is situated about 100 miles south of Itat on the road to Chibakee. It

<sup>20</sup> C. W. Purington, "The Gold Mines of Siberia," *Engineering Magazine*, September, 1901.

<sup>21</sup> For further description of the apparatus in use, and for comments on the Siberian methods, see Levat, DeBatz, and Purington, *op. cit.*



extends into the eastern slopes of the Altai, heading about 50 miles to the west of the Itat post-road, on the divide which forms the natural and political boundary between the Achinsk mining-district and the lands belonging to the Private Cabinet Land of the Tsar. The valley at its lower, or eastern, end is about four miles wide, and maintains this width for 20 miles up the river. The Sorela itself is a tributary of the Chorni Oos, and is nearly 75 miles long. It has the characteristic of Siberian rivers of retaining the same width for great distances. For 30 miles it continues about 100 ft. wide. After the first five miles it flows through an immense gravel-flat which is nearly level. The road leading to the placers, at its upper end, crosses the Sorela from 20 to 25 times by means of fords, which are often difficult to pass through, on account of the floods. The mountains at the head of the Sorela, rising to heights of 3,000 ft. above the general level of the river, consist of diorite, in rounded domes. Where exposures were visible, slates of a graphitic character, and fine-grained porphyries, were also seen. The gravels of the Sorela valley are more rounded in character than the gravels farther to the south, and, in fact, the whole nature of the region shows the existence of older geological conditions. There has been a greater amount of erosion, and a more extensive scooping out of the valleys.

In the lower part of the valley the cliffs on either side consist of limestone, probably of Devonian age. The limestone appears, from its dip, to lap around the base of the diorite-bosses, which compose the high mountains to the westward.

More than \$5,000,000 in placer-gold has been taken from the placers of the upper Sorela in the early days of Siberian gold-mining. At present no mining is done except the reworking of old tailings on a comparatively small scale. Selected portions of these tailings carry 60 cents per cubic yard. The gravel consists of diorite, limestone and volcanic rocks.

It is the extreme upper end of the Sorela valley, above the old placer-workings, which is most interesting. Here the stream becomes a mountain torrent, and at the base of a great amphitheater, which forms the head of the valley, as many as five distinct tributary torrents unite to form the main stream. It is estimated that, in the summer, 3,000 miner's inches are available for water-power, and 2,000 in. in the winter. By

systems of flumes to carry the water to a central point, a large amount with several hundred feet of head would be available for power at the juncture of the fingers of this remarkable stream. On the high slopes of the bare mountains above lie great banks of perpetual snow. Hollow basins, formed by glaciers of a local character, are found in the upper valley, and a small lake occupies one of these, into which falls a cascade from a torrent above. The upper 1,000 ft. of the mountains are bare, except for thick beds of reindeer-moss. Below this commences a thick timber-growth of pine and spruce, with a lesser amount of larch.

The rock at the head of the Sorela valley has lately been prospected for quartz-veins, due to the accidental finding of gold-bearing quartz four years ago.

The reindeer-moss, which effectually hides the outcrops above the timber, has been stripped from a considerable area, revealing the presence of quartz-veins of remarkable width. These veins have been for the most part found on the north side of the basin, and appear to form parts of a regular system, striking northwest. One of them, which was uncovered by an open cut for 200 ft. down the mountain-side, showed an outcrop of 10 ft. wide for this entire distance. In places, the distance from wall to wall was 20 ft. of what appeared to be solid quartz. A poor mill of the Chilean type was treating the quartz, and 500 tons of ore which had been treated was said to have yielded \$18 per ton without sorting. This statement I had no opportunity of verifying.

To the southwest of this vein, and parallel to it, another vein had been uncovered, showing for a width of from 4 to 6 ft., and exhibiting free gold in the majority of pieces broken from the outcrop for a length of 500 ft.

The number of lodes in the Sorela valley showing free gold on the outcrop, which have already been uncovered, is unusual. Why the lodes have remained for so many centuries unexploited is indeed hard to explain. Owing to the peculiar topographic conditions, the permanent water-level in the region, and consequently the refractory portions of the veins, must lie at a considerable depth. It is, therefore, all the more difficult to understand why attempts have not been made before to extract gold from the free-milling outcrops and upper portions of the veins.

## Mineral Resources of British India.

BY SARAT C. RUDRA, CALCUTTA, INDIA.

(New York Meeting, October, 1903.)

### INTRODUCTION.

GEOGRAPHICALLY, British India includes India or Hindustan proper, Burma, and a part of Beluchistan.

In ancient times India occupied a very important position among the producers of minerals and metals; in fact, as late as the middle of the seventeenth century, it contributed a very considerable quota of the mineral and metal supplies of the world. The branches of the industry worthy of special note are gold, silver, lead, antimony, copper, iron, diamonds and other precious stones. The famous "Damascus steel" was originally manufactured in India, Damascus being only a mart; and it is interesting to note that even at the present time this product is still made in India. Prior to the discovery of diamonds in Brazil in 1728, India was the sole producer of this important mineral; and while no diamond-mines are now systematically worked, fine specimens are occasionally reported to be found in river-beds and in alluvial diggings.

For ages, nearly all of the minerals of present commercial value have been produced and worked in India, and many of the deposits may still play an important part in the world's mining and mineral history.

Despite the patient work of the ancient Indians, the limited knowledge they possessed of pumping and ventilation precluded the possibility of attaining any great depth in mining. Under special conditions, shafts have been sunk to 600 ft., but the average depth was much less. As a result of this limitation, the ancient miners confined their work to the available ores in sight, leaving large bodies of ore intact, which can be profitably worked with modern methods of mining. As an evidence of this statement may be mentioned the mines of the Kolar gold-fields in Mysore, which have reached in some shafts a depth exceeding 2,400 ft. Traces of ancient work in these

mines are seen at depths not greater than 600 ft. from the surface.

Dr. Ball, a late member of the Geological Survey Department of India, has published an excellent book on the economic mineral resources of India,<sup>1</sup> and the Geological Survey Department has now in preparation a revised and enlarged volume on the mineral resources of the country.

It has been a matter of surprise to many who have made a careful study of the possibilities of successful mining in India, that British capitalists have been so slow in taking advantage of the knowledge of the ancient mines scattered throughout the country. Indeed, it seems to be the repetition of a fatality often attending the development of mining-fields. Sometimes a mining-field, although rich and promising, will not attract capitalists because the country is not in fashion,—a condition which may be said to exist in India at the present time. However, a few more examples like the Kolar gold-fields will sooner or later bring a change of opinion, and Indian mines will be in demand. What more encouraging fact can be placed before the public than the result obtained by the group of Kolar mines, not more than a dozen in number, which has produced in recent years more than 500,000 oz. of gold annually, and has placed India seventh in the list of the gold-producing countries of the world.

Apart from gold-mines, hardly any other metalliferous deposits have yet attracted foreign capital to India. It is true that spasmodic efforts have been made to open up one or two copper-mines within the last 60 years, but the failure attending these efforts (owing to bad management) has set back the copper-smelting industry. High-grade copper-ore is known to exist in many parts of India. For instance, a belt of country 80 miles long, and within a few hours' ride by railroad from Calcutta, is known to contain excellent high-grade copper-ores; and the large slag-heaps in this belt testify to the quantity of copper extracted by the ancients with their crude methods.

Throughout India there is an inexhaustible supply of high-grade iron-ores; and the diamond-fields, though seemingly exhausted, may again become the wonder of the world.

Dr. Ball in the introduction to his excellent book,<sup>2</sup> referring

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<sup>1</sup> *A Manual of the Geology of India*, by Dr. V. Ball, Calcutta (1881).

<sup>2</sup> *Op. cit.*

to the mineral deposits of India, remarks with singular force and truth as follows :

“The habit of speaking contemptuously of the mineral deposits of the country having grown up so strongly, the question of the value of particular deposits is now far too often unfavorably prejudged. Speaking generally, the value of the majority of deposits is relative to external circumstances. Were India wholly isolated from the rest of the world, and were her mineral productions protected from competition, there cannot be the least doubt that she would be able, from within her own boundaries, to supply very nearly all the requirements of a highly civilized community, in so far as the mineral world is concerned. But the consumer would probably have to pay more than he does at the present day.”

### PHYSICAL GEOGRAPHY.

India-proper is broadly divided into the three sections :—

1. Peninsular India, which incudes all that part of the country traversed by the Satpura and Vindyan ranges of hills, and the country lying south of them. The geological age of rocks in this section belongs to the Lower Paleozoic and older formations.

2. Extra-Peninsular India, which includes all that part of the country which forms the base of the Himalayan Mountain ranges and the contained valleys. The geological age of rocks here corresponds to the Pliocene.

3. Indo-Gangetic Plains, which connect Peninsular and Extra-Peninsular India, and are watered by the Indus and Ganges rivers and their tributaries. The formation in this region is of recent origin.

Peninsular India is drained in its northwestern part by the Nerbuda and Tapti rivers, which flow into the Arabian Sea. The other great rivers, Mahanadi, Godavari, Krishna and Kavery, have a drainage area to the east, flowing into the Bay of Bengal.

The mountain systems are the Western Ghats (or Sahyadri), which extend south from the valleys of Tapti along the coast, terminating in Mysore and Coorg. The highest point in this range is Mohaballeswar, 4,540 ft. These hill-ranges slope to the east, the rise of the Ghats being sudden and precipitous.

In the East Coast, a range of hills, whose chains are not as continuous as the Ghats in the west, extend north from the Nilgiri Hills and terminate in the Lake Chilka district. Some of the hills in this section are 8,000 ft. high.

The rock-formations in the northern part are horizontal and basaltic, and are replaced south of lat.  $16^{\circ}$  by metamorphic rocks.

The central portion of the country comprising the Deccan Plateau has an elevation of from 2,000 to 3,000 ft. Mysore, Hyderabad and adjoining countries are situated in this high table-land.

Judging from the contour of Peninsular India and the position of its rock-formations, it becomes evident that this part of the country has not undergone great geological change. The basaltic rocks found in west Satpura are horizontal or very nearly so. Central Satpura contains hills formed partly of trap-sandstone and metamorphic rocks. Panchmari, however, has formations of Mesozoic sandstone; farther east, horizontal traps reappear; and still farther east, north of Bilaspur, there is a broad expanse of undulating ground at a lower level; and, in Chota-Nagpur, a plateau of metamorphic rocks capped by horizontal trap and laterite appears.

There are strong indications that the whole of Peninsular India was at one time a high land, and that denudation has played a very important part in making its present configuration. Further, the remarkable absence in Peninsular India of any evidence of disturbance in late geological times distinguishes it from the rest of Asia.

The area of disturbance in the Aravali range of hills corresponds with the direction of the range itself. The formations, which belong to the transition-period of great antiquity, even older than the Vindyan, are non-fossiliferous.

The mountains of Extra-Peninsular India,—Sinde, Punjab, Himalayas, Assam, Chittagung and Burma, and elsewhere,—have formations much disturbed and contorted. The disturbance has greatly affected the rocks of later origin; but some exceptions to this statement may be observed in parts of Assam.

#### GEOLOGY.

The classification of the geological formation of Peninsular India is given in Table I., and the corresponding relation with European deposits in Table II. as published in the reference given below.<sup>3</sup>

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<sup>3</sup> *Manual of the Geology of India*, Ball, Part I., pp. xiii and xiv.

TABLE I.—*Classified List of Geological Formations in Peninsular India.*

		Approximate Maximum Thickness.
		Feet.
RECENT AND POST-TERTIARY.....		700 unknown (from boring).
CENOZOIC.	TERTIARY :	
	Milliolite of Kattywar, Pliocene, Miocene and Eocene (Numilitic beds of Kutch and Guzrat, sandstone, clays and lignites of the west coast), Travancore and Ratnagiri, Cuddalore sandstone and high-level laterites.....	2,700
	Deccan Trap.....	6,000
MESOZOIC.	Marine Cretaceous :	
	Arialur, Trichi. group, Bagh beds, Neocomian of Kutch.....	3,000
	Marine Jurassic .....	6,000
PALEOZOIC.	Gondwana System :	
	{ Upper.....	11,000
	{ Lower.....	13,000
	Vindyan Series :	
	Upper—Bhandara, Rewah, Kimur.....	12,000
	Lower—Kurnul, Vhima, Sone, Somri.....	2,000(?)
	Transition or Sub-Metamorphic :	
	Upper—Gwalior, Kadapah and Kaladgi Series.....	20,000
	Lower—Bijawar, Champanir, etc.....	Unknown.
	Metamorphic.	

TABLE II.—*Relation of Indian to European Geological Deposits.*

		Peninsular Rocks.	Indian.	European.			
PALE- MESO- CENO- OZOIC. ZOIC. ZOIC.	{	{	High Level Laterite.....	Numilitic.	Middle Eocene.		
			{	{	.....	.....	Lower Eocene.
					Upper Deccan Trap..	Arialur.	Upper Chalk.
					Middle Deccan Trap.	Trichmopoly.	Lower Chalk.
					Lower Deccan Trap..	Bagh Beds.	Lower Chalk.
					Intertrepan.....	Uthtur.	Upper Greensand.
					Jabalpur and Kutch.	Umia and Katrol.	{
					Mehadeva, Rajmahal	Chari and Pachbum.	
					Panchet.....	.....	Triassic.
					Damuda.....	.....	{
					Talchir.....	.....	

## MINERAL AND METAL PRODUCTION.

The following statistical notes have been taken largely from the two publications mentioned below;<sup>4</sup> and Table III., at the

<sup>4</sup> *Statistics of the Mineral Production of India in the Ten Years 1892 to 1903*, Calcutta, 1903, and *Mines and Quarries, General Report and Statistics for 1901*. Part IV., Colonial and Foreign Statistics, C. LeNeve Foster, London, 1903.

end of statistical section of this paper, containing the quantity and value of the individual output for the years 1890, 1900 and 1901, has also been compiled from these sources.

In order to give the proper geographical importance to each of the localities mentioned in the following statistical notes, small capital letters are used to denote PROVINCES, and italics to denote *Districts*.

*Antimony*.—The extensive use of antimony in India is to anoint the eyelids of children as a protection from the influence of the weather, and to pencil the eyelids of women. It is also used medicinally, and as an alloy in the manufactory of brass utensils for household purposes. The localities where antimony minerals occur are: MADRAS, *Bellary*; MYSORE; *Vizagapatam*; HYDRABAD (Deccan); *Hazaribagh*; *Ajmir*; BELUCHISTAN; *Beyla*, *Sekran*, *Quetta*; CASHMERE; PANJAB; *Lahoul*; *Shigri*; LOWER BURMA, *Tenasserim* and *Amherst*.

The most remarkable lode is that which has been rediscovered in *Shigri* by Messrs. Calvert, Younge and Major Hay. There are two lodes, the upper being of enormous size, although its dimensions have not yet been accurately ascertained. The lower one, however, is nearly 40 ft. wide. Attempts were made to open these mines by Mr. Calvert, but the enterprise was a failure, owing to the inaccessibility of the deposit, which was at an altitude of 14,000 ft. and had no road communications. This enormous lode contains 20 ft. of antimony sulphide (stibnite), the exposed parts containing cervantite and kermesite.

In *Shigri* there are six distinct lodes of an average yield of 50 per cent. of metallic antimony.

In PANJAB at *Kangra* and *Spiti*, antimony is found in pyrite. The ores found in *Peshwar* and *Bijaur* are good. According to Major Baden-Powell,<sup>5</sup> antimony is found at Simur near Simla.

The antimony sold in Cashmere is of superior quality and probably comes from *Jammu*.

*Arsenic*.—The principal supply of arsenic, in India, is from imported ores, chiefly from the adjoining countries, Swat and Kasghar in Central Asia. The Bhutias, however, market a

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<sup>5</sup> *Punjab Products*, p. 11.



yellow orpiment, and the Burma production of red orpiment is always for sale in the bazaars.

*Bismuth.*—Although bismuth has not yet been found in India in a metallic form, traces of it are found in the copper- and lead-ores of BENGAL, *Hazaribagh* and *Singbhoom*; PANJAB, *Mundi State*; NEPAL and BURMA. The Nepal ore may properly be called bismuth ore for the reason that it contains 24.6 per cent. of bismuth and only 14.4 per cent. of copper.

*Chromium.*—Veins of chromite occur in serpentine or magnesite rock-formations, in Peninsular India near Salem, and in Extra-Peninsular India, near *Spiti* in the Himalayas.

*Coal.*—The mining of coal may be reckoned as the most important and valuable of the mineral resources of India, and although the production has not exceeded in value that of gold, the coal-industry in India ranks first. The rapid increase in the production of coal within the last few years indicates the active expansion of industrial developments. As the export trade is as yet very insignificant, Colombo and Singapore being the two great foreign markets, the total quantity shipped annually from the country does not exceed a million tons.

There are approximately 264 operative mines, of which nearly 240 are in BENGAL, the rest being in CENTRAL PROVINCES, HYDRABAD, PANJAB and BELUCHISTAN.

The Singarini coal-mines of HYDRABAD rank next in importance to those of BENGAL. Next to Singarini are the Makum mines in northeastern ASSAM.

The coal-bearing area of India is very extensive. In this respect I believe that India has the third largest field in the world; China and the United States being second and first respectively.

Some of the coal-seams in India are of extraordinary size. For instance, in ASSAM the Makum coal-bed near Debrughur is more than 100 ft. thick. In BENGAL, at *Jheria*, one of the seams now worked exceeds 40 ft. in thickness. In addition to many beds of these unusual sizes, the average thickness of a number of seams usually worked lies between 6 and 12 ft. The present localities where mining is carried on, are—BENGAL, Raniganj, Jheria Seetarampore, Giridi, Daltongung, etc.; HYDRABAD (DECCAN), Singarini; UPPER ASSAM, Sadyia, Lakhimpur; CENTRAL PROVINCE, Mohpani and Warora; and CENTRAL INDIA AGENCY, Umaria.

A few years ago, India imported a considerable quantity of coal, especially for the BOMBAY PRESIDENCY, but for the last four or five years Bengal coal-mine proprietors are actively securing all the available markets in India, notably Bombay and Karrachi, on the West; Madras and Colombo on the South; and Burma and Singapore on the East. The rapid increase in the consumption is due not only to extended export trade, but also to the rapid expansion of Indian railways and the various industrial and commercial establishments. The output of coal in India during 1890 was 2,168,521 tons, which was increased to 6,118,652 tons in 1900.

The coal-beds in India seem to correspond with those in England, nearly all the coal now worked being of the Permian-Triassic age. Most of the coal-fields are situated in BENGAL, and 240 mines near Calcutta are now operative. It is called the "Black Country" and may be appropriately designated as the Pennsylvania of India, but with this difference that almost all of the coal is soft, anthracite being found only in one place near Giridi, in *Hazaribagh*.

The following comparison of the quality of the best coal found in the Karharbari mines, Giridi, with Welsh coal may prove of interest.

	Specific Gravity.	Ash.	Fixed Carbon.	Sulphur.	Volatile Matter.	Authority.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	
Karharbari Mines:						
Lower seam.....	1.35	9.15	66.84	0.42	24.00	Dr. Saise. Report for the Navy.
Welsh coal.....	1.31	3.68	82.06	1.59	13.66	

Analyses of two samples of coal from the Singarini mine (lat. 17° 30' 30'', long. 80° 20') are as follows:—fixed carbon, 62.4 (66); volatile matter, 32.6 (23); ash, 15 (11) per cent.

An analysis of coal from Makum in ASSAM is:—carbon, 60; volatile matter, 36.2; and ash, 3 per cent.

The Singarini mines, now producing more than 50,000 tons a year, have partially been successful in meeting the industrial requirements for railways, mining, cotton industries, etc., in the Nizam's territory.

*Cobalt*.—Cobalt minerals have been found only in three or four places in Peninsular India, notably, at *Khetri* and *Jeypore*

(Babai, Bagor, Singhana), in RAJPUTANA. In Extra-Peninsular India it is found in NEPAL and BURMA. The principal use of cobalt in India is in the enameling of metals. At present there is no domestic production of ore.

*Copper.*—In Peninsular India, copper-ores are found in old crystalline and metamorphic rocks as well as in several groups of transition rocks, while in Extra-Peninsular India they are mostly found in highly metamorphosed rocks, a difference in condition which is not easily understood. The most common occurrence below the surface is copper sulphide, which becomes altered into carbonates and oxides as the outcrop is approached. The prevailing idea is, that true copper-lode formations do not occur in India, the deposits being sparsely disseminated, or locally concentrated in bunches and nests. This opinion has no doubt been formed with imperfect knowledge, for the reason that little or nothing has been done to ascertain whether copper-ores exist at greater depths. The knowledge of the formations are mostly derived from a few scratchings that have been made in connection with mines in CHOTA-NAGPUR, *Kumaun* and SIKKIM, the depth in no instance exceeding 250 ft. Therefore, it is premature to deduce conclusions without a more thorough search.

Copper has been found in CHOTA-NAGPUR, at *Singbhoom*, *Manbhoom*, *Hazaribagh*; in RAJPUTANA, at *Jeypore* and *Khetri*; in NEPAL and SIKKIM; at *Kumaun* (in the vicinity of Almora); in CASHMERE and *Kulu*, and in UPPER BURMA and LOWER BURMA.

I have already mentioned in the introductory notes of this paper the large belt of country containing copper-ore in *Singbhoom* near Calcutta. Since 1852 four attempts have been made to open a mine; but, notwithstanding the good facilities of labor and capital, they have utterly failed. These results were not due to poor ore, because but very little work was done; the deepest shaft reached only 110 ft. and the quantity of ore raised was about 4,000 tons. The failures of the Hindustan Copper Mining Co. and its successors are attributable to incompetent management; and the development of this copper-belt country, and indirectly of India, has been hindered by their unsuccessful efforts.

The copper-ores of Peninsular India occur in metamorphic rocks and in transition rock-groups. In Extra-Peninsular

India they are found mostly in highly metamorphosed rocks. The places in India where copper-deposits occur are grouped in the subjoined list:—

MADRAS PRESIDENCY, *Trichinopoli*, *Bellary*, *Cuddapah*, *Karnul* and *Nellore*. (In the mines in these districts the minerals are chrysocolla, chalcocite, malachite, ferruginous cupric oxide and copper pyrites.)

HYDRABAD, *Ramgir*.

BENGAL, *Doeghar* (peacock ore).

CHOTA-NAGPUR, *Manbhoom*, *Purda* and *Kuliampur*, and *Singbhoom*.

Evidence of large mining and smelting operations by the Seraks (lay Jains) are found in a great belt of the country nearly 80 miles long, commencing with *Lopso* in *Lohardaga* and extending to *Kamerara* in *Midnapur*. Special mention should be made of the mines at *Landu*, *Jamjura* and *Rajdoha* all in *Singbhoom*, the minerals being copper pyrites and carbonates. Also of the oxide and sulphide ores of the abandoned *Burragunda* mines in *Hazaribagh*.

CENTRAL PROVINCE—*Raipur*, *Jabalpur*, *Narsingpur*, *Chanda*, *Rewah* and *Bundelkhand*.

RAJPUTANA—The principalities of *Alwar*, *Bharatpur*, *Jeypore*, *Khetri*, *Ajmir*, *Udaipur*, *Bundi* and *Bikanir* have copper-ores which can be worked commercially. Of these, *Khetri* undoubtedly possesses the best mines, which have long been worked with profit.

BELUCHISTAN, *Las*.

BOMBAY, *Dharwar*.

PUNJAB.—In the *Salt Range*, *Simla*, *Kulu* (also the States of *SIRMUR* and *CASHMERE*), are copper-ores which are said to be of good quality and of high percentage of metal. A few years ago, a piece of native copper weighing several pounds was found in *Cashmere*, but a thorough search of the place where it was found failed to reveal the matrix from which it came.

UNITED PROVINCE.—In *Kumaun* and *Gharwal* are remains of several ancient mines. The prospects of mining and smelting are reported to be satisfactory; but hitherto success has not attended any of the enterprises. At *Darjiling* and *Western Duars* (*Boxa*), the outlook for mining copper-ores is promising.

BURMA.—In the *Arakan* and *Tenasserim* Divisions, copper-ore is known to exist, but no attempt has been made to investigate the prospects thoroughly.

*Corundum*.—For centuries this mineral has been worked in India for useful and ornamental purposes. In Peninsular-India it is found mostly in Southern India, but there is an important deposit in CENTRAL INDIA AGENCY at *Rewah*. CHOTA-NAGPUR has very fine deposits, but hitherto no attempt has been made to open them up. There are a few deposits also in CENTRAL PROVINCES.

The principal localities in Extra-Peninsular India are in CASHMERE, where fine emeralds and sapphires are frequently found. However, BURMA may properly be called the home of the corundum family:—viz., ruby, emery, sapphire, topaz, emerald, oriental amethyst, adamantine, spar, etc.

Many other precious and ornamental stones,—notably, agate, moss-agate, cat's-eye, Balais-ruby,—are found in India and Burma.

The export trade of corundum within the last few years has been increasing in India.

*Diamond*.—This pure crystalline form of carbon has played a very important part in the history of ancient and modern India, and references to this mineral are found in many of the ancient Sanskrit writings of India, in which names of localities where diamonds were found are also given; although to recognize some of these localities is rather difficult owing to changed nomenclature. Rajah S. M. Tagore, in the publication named below,<sup>6</sup> has tried to establish the identity of these localities, with fair success. The following are the names of the places referred to in ancient Sanskrit literature: Haima (Himalaya); Matanga (*Krishna* and *Godavari*, in Golconda); Surashtra (Surat); Paunda (probably CHOTA-NAGPUR); Kalinga (between Mahanadi and Godavari); Kosala (Oudh or Berrar?); Venganga (Weinganga); and Saubira (tract between Sirhind and the Indus river).

Marco Polo mentions<sup>7</sup> Matanga (Telingana) as Mutfli, the latter name being derived from the port called Matupalle.

Karl Ritter suggests in his book<sup>8</sup> that the Arabs and the Phœnicians had a regular trade in diamonds with India. He also found evidences that the trade existed in the Solomonian and Mosaic periods. In addition to Marco Polo, much infor-

<sup>6</sup> *Moni Mala*, 2 vols., Calcutta (1879).

<sup>7</sup> *Travels of Marco Polo*, edited by Col. Yule, vol. ii., London (1871).

<sup>8</sup> *Erskunde Asien*, vol. vi., p. 343.

mation regarding diamonds in India has been published by Tavernier<sup>9</sup> and Fitch.<sup>10</sup>

The name "Golconda diamonds" was not derived from the town of Golconda near the city of Hyderabad, but was used for the product obtained from the extensive regions comprised of the provinces watered by the Krishna and Godavari rivers.

Besides Golconda, the other localities in the same neighborhood where diamonds were formerly mined, are *Cuddapah*, *Bellary* and *Karnul*.

The diamond-producing fields in India may be divided roughly into three sections:—

1. Southern: Golconda or Telingana, including the five modern districts of *Cuddapah*, *Karnul*, *Bellary*, *Krishna*, *Godavari* (Bhadrachalam).

2. Middle: Includes the large tract of country between the rivers Godavari and Mahanadi. In this section diamonds are still found near Sambhalpur and Warragurh. There are also two or three localities within CHOTA-NAGPUR where diamonds are occasionally found in river-beds.

3. Northern: Includes the country known as *Bundelkhand*, in which is situated the *State of Panna*. In the country surrounding *Panna*, diamonds occurring in place are being mined regularly.

In northern India diamonds occur in Rewah group of the Upper Vindyan formation, and in lower India, MADRAS PRESIDENCY, in the Kurnul (Silurian?) formation. The geological strata of northern and southern India as described by Dr. Ball<sup>11</sup> are as follows:

Northern India.	Southern India.
Upper Vindyan Series.. { 1. Bhaurer group..... 2. Rewah group (diamond) 3. Kaimur group.....	} Absent. (Known as the Kurnul formation.) 1. } Khundair shales and 2. } limestone. 3. Panceun quartzite. 4. Jamalnadgu shales and limestone. 5. Banaganpally (diamond)
Lower Vindyan Series.. { 1. Tirhowan limestone..... 2. Palkoa shales..... 3. Dalchikur sandstone... 4. Semri shales and lime- stone..... 5. Semri sandstone.....	1. } 2. } 3. 4. 5.

<sup>9</sup> *Voyages*, Paris, vol. ii. (Translated by V. Ball, *Travels in India*, 1889.)

<sup>10</sup> Selections from *The Records of the Bombay Government*, viii., p. 7 (1853).

<sup>11</sup> *Op. cit.*

So far as known the occurrence of diamonds at *Panna* is limited to the Rewah group, being found in places in a conglomerate rock, and in alluvial and superficial deposits. The Rewah bed extends over a large area, but no search for diamonds has been made elsewhere than at *Panna*. Diamonds are sometimes found included in pebbles.

In CHOTA-NAGPUR and *Sambhalpur*, diamonds are found in river-beds, from which they are obtained by some low-caste tribes in the following primitive manner: After a rainy season, the Mahanadi river near the town of Sambhalpur becoming low, a large number of these tribes begin to dam the north channel of the river between an island and the bank. Later, when the river gets still lower, the gravel included in this dam is collected and carried to a dry flat place, where it is washed for gold and diamonds. The quantity of gold obtained is very small indeed, and may be barely enough to pay for the daily meals of one person, but sometimes the washers are rewarded by the find of a good-sized diamond, which may keep a family in ease and comfort for years. The diamonds found in this river-bed have evidently been brought down from a higher source, perhaps from the headwaters of the Mahanadi river, and a thoroughly equipped prospecting expedition would doubtless find the matrix bearing the diamonds.

In the southern Golconda regions, diamonds are found in gravel-beds composed of rolled stones of various sizes, intermixed with mud. The pebbles are ferruginous, schistose sandstones or sandstone-conglomerates, and include also, quartz, chert, jasper, claystone, porphyry, feldspar crystals, blue jasper veined with iron oxide, red jasper and quartz crystals.

Reports of the finding of diamonds near Simla in the Himalayas are very interesting. The older Palaeozoic rocks somewhat resemble the southern Kurnul (Silurian?) formation. It is, therefore, likely that diamonds will also be found in this locality.

The following list is comprised of the localities in which diamonds have been found:

MADRAS PRESIDENCY.—*Cuddapah* (Chenney, lat.  $14^{\circ} 34'$ , long.  $78^{\circ} 51' 30''$ ), a village near the town of Cuddapah. A number of square pits in the gravel-formation show the great extent of the old workings. Attempts to open up these deposits were made in the year 1869, but the results were unsatisfac-

tory, due to the small size and inferior quality of the diamonds obtained.

*Oblampally* (lat.  $14^{\circ} 34'$ , long.  $78^{\circ} 51' 55''$ ).—This field is one of the latest ones opened. The crystalline forms of the diamonds found in this district are much disguised, but the brilliancy when wet is of the highest degree.

*Bellary*.—At Munimadadu (lat.  $15^{\circ} 17'$ , long.  $78^{\circ} 2' 30''$ ), the mines, formerly productive and famous, were abandoned in 1813. Diamonds occur also at Wajarakarur (lat.  $15^{\circ} 2'$ , long.  $77^{\circ} 27'$ ), and Gutti (lat.  $15^{\circ} 7'$ , long.  $77^{\circ} 42'$ ).

*Kurnul*.—At Banaganpally (lat.  $15^{\circ} 18' 30''$ , long.  $78^{\circ} 16'$ ), 87 miles SSE. of the town of Kurnul, the workings are in rock-beds. At Ramulkota (lat.  $15^{\circ} 34'$ , long.  $78^{\circ} 3' 15''$ ), the workings are in rock-beds and in alluvial. Diamonds occur also at Timæooram (lat.  $15^{\circ} 32' 30''$ , long.  $78^{\circ} 6' 30''$ ), 6 miles SSE. of Ramulkota.

In addition to these localities may be mentioned those of Yembye, Bryanpullygooramankonda, Goodypand, Baunoor, Devanoor, Shaitancotah, Doomurroh, Tandrapad, Buswapoor.

*Krishna and Godavari*.—Kollur (Coulour mentioned by Tavernier) (lat.  $16^{\circ} 42' 30''$ , long.  $80^{\circ} 5'$ ) is supposed to be the home of the famous Koh-i-Nur and the Great Mogul diamonds. Tavernier leaves record, that at the time of his visit to Kollur nearly 60,000 men, women and children were engaged in digging diamonds at that place. At Bhadrachalam, north of the Godavari river, not long ago, an accidental find in stream-gravel of a diamond valued at \$7,000 was reported.

CHOTA-NAGPUR.—About 150 years ago, Simah (lat.  $23^{\circ} 35'$ , long.  $84^{\circ} 21'$ ), on the Sunk river, flourished as a town, but it is now in complete ruins. Systematic diamond-washing was formerly carried on with profit in the streams near the village, and it is possible that diamonds in place will be found in this neighborhood. There are ample evidences that diamonds occur in the beds of the Ebe, Koel and Sunk rivers. A study of this part of Bengal shows the possibility of developing a great diamond-field in the table-land of CHOTA-NAGPUR. The streams and rivers draining the district flow either to the Ganges on the north, or to the Mahanadi and Hugli rivers on the south and southeast. Diamonds are still found on both slopes of this table-land, and it is, therefore, not unreasonable to suppose that a careful search will be rewarded.



CENTRAL PROVINCE, *Sambhalpur*.—Gold- and diamond-washing were carried on actively 60 years ago at a spot not far from the town of Sambhalpur.

*Chanda, Wairagarah*.—At Beiragarh (lat.  $20^{\circ} 26'$ , long.  $80^{\circ} 10'$ ), 80 miles SE. of Nagpur, the prospect of finding diamonds is very good.

*Bundelkhand, Panna* (lat.  $24^{\circ} 43' 30''$ , long.  $80^{\circ} 15'$ ).—The mines in this locality are still being worked and the field is very interesting. This is the only known locality in India where diamonds are found in place. Other sections in this field are Kamariya, Birjpur, Udesna, Sakenya and Saya Lachmanpur.

*Epsom Salt*.—This mineral is found in PUNJAB, at the Lower Spiti and Mayo mines.

*Gold*.—The application of modern mining methods and metallurgical processes to the extraction of gold in India is yet in its infancy. Most of the present output, which during 1902 exceeded 500,000 oz., is obtained from the mines of MYSORE in the Kolar fields. A few mines are worked also in HYDRABAD and BURMA, but the development has not grown to any extent. The value of the gold-output in Mysore during 1892 was 163,180 oz., valued at \$3,300,000, while in 1901 it had increased to 529,782 oz., valued at about \$9,000,000. The gold-fields in HYDRABAD, BURMA and ASSAM have not yet proved a lasting success.

In the early centuries the gold produced in India was obtained both from alluvial washings and from reef-mining. The latter class of work, however, has been practically suspended during the past three centuries up to the time that the mines in the Kolar gold-field were opened. The washing of gravel to obtain gold has been carried on for years by the Jhoras or Dhoras, but the exhaustion of the deposits, due to centuries of working, has reduced this formerly important branch of the industry to insignificant proportions. At the present time the washers barely manage to eke out a precarious living. For two or three months after the close of the rainy season, the washing of gravel from the bed of a stream is the only source by which a washer obtains a few grains of gold daily.

In connection with gravel-washing it is interesting to note the manner in which gold is deposited year after year in hollows in the rocks and at other convenient places in the bed of

a stream, and it is well within the range of probability that a thorough and systematic exploration would show the original source of the gold now present in the stream-gravels.

Although at the present time gold-mining is limited to southern India, careful prospecting is certain to develop other places which will become important factors in the future output of the country. The brief notes of the places named in the following list present a clue to those sections of India in which prospecting work promises good returns.

**BENGAL (CHOTA-NAGPUR).**—*Singhbloom*, Kapargadi Ghat, Kamrara, *Kherswan*, *Seraikela*, Porahat, Sonapat (found in reef and river-beds); *Manbhoom*, near Bhagmundi (in granitic gneiss); Supur near Namra (in alluvial and river-washing). There are other places wherein the gravel specks of gold are invariably found by washing. The rock-formation at these places is sub-metamorphic.

*Jashpur State*.—At Pharsabet (lat.  $20^{\circ} 30'$ , long.  $83^{\circ} 55' 33''$ ) and other localities, gold of superior quality and quantity is found in both alluvial- and rock-formations; and the semi-civilized people using crude methods of gravel-washing produce a small quantity of the precious metal every year. This State offers a good field for prospectors.

*Udaipur-State*.—The prospect of finding a rich gold-field in Udaipur State is very assuring. Attempts have been made to open up some of the mines that were formerly worked by the natives, and while the results were satisfactory from the financial point of view, the unhealthiness of the country was fatal to the mine managers. This objectionable feature has been removed by the extension of railroads and the clearing out of the jungles, and the necessities and comforts of life are now procurable with comparative ease. Many places on the Koriyah river are known to be gold producing, notably Kamhar (lat.  $22^{\circ} 35'$ , long.  $83^{\circ} 18' 15''$ ), Sangul and Bharari. There are also many places on the Mani river at which gold has been obtained.

**LOWER BENGAL.**—*Midnapur* and *Bankura*. Gold has been obtained by washing the alluvial soil.

**ORISSA.**—*Daenkanal*, *Keonjhore* and *Lahara*. Gold occurs in nearly all of the streams in these districts, and the prospects of reef-mining is very favorable, especially in *Daenkanal* and *Keonjhore*.

CENTRAL PROVINCE.—A number of localities in this province give good prospects of obtaining gold by washing; notably *Sambhalpur* (Tahud); *Bilaspur* (Sonakhan); *Raipur* (Ragim); *Bhandara* (Ambagh and Thirora). Gold-bearing sand is also found in Maroo river. *Chanda* (stream-gold); *Balaghat* (Lanji, Panchera and Mau); *Jabalpur* (stream-gold); *Nagpur* (stream-gold, country-rock metamorphic); *Waradha Sagar*, *Droh*, *Seoni*, *Upper Godavari* (Marigudene and Bastar).

RAJPUTANA.—*Gurgaon*.

BOMBAY.—*Dharwar*. Prospecting and mining have been vigorously carried on at Soordoor, Dhona Dhambal, Harti and Moolgoond. These localities are favorably reported upon by the Geological Survey Department. *Belgaum*, *Kaladgi* and *Katwyar* districts also contain gold-bearing deposits.

PANJAB.—*Bannu*, *Peshawar*, *Hazara*, *Rawalpindi*, *Jhillum*, *Kangara*, *Umballa*, *Patilama*, *Cashmere* (alluvial) and *Ladak* (in western Thibet, belonging to the Marajah of Cashmere). In *Ladak* many rich mines in veins and rock-formations are supposed to exist. Mines are worked at Kio (lat.  $34^{\circ}$ , long.  $37^{\circ} 18'$ ).

UNITED PROVINCE.—The river-beds of *Kumaun*, *Gharwal* and *Mooradabad* are auriferous. *Champaram* contains auriferous rocks, and the washing of river-gravel is carried on regularly. The gold has doubtless been derived from rocks higher up in *NEPAL*, although there is no report of gold-washing in *NEPAL* or *Sikkim*.

BHUTAN.—This country, occupying the northeastern corner of the Himalayas, is reported to be very rich in mineral resources. The country is no way connected politically with British India and no prospecting has been carried on. All that is known concerning its gold-deposits is that the Bhutias, on their yearly trading mission to Bengal, bring quantities of gold-dust for purposes of exchange. There is no doubt that the country is very rich in mineral products.

ASSAM.—*Darrang*, *Sibsagar*, *Lakhimpur*, *Khasi Hills* and *Monipur* have auriferous river-beds from which gold is extracted.

TIPPERAH.—This section of the country is south of Assam and in eastern Bengal, and is perhaps the "Tipra" mentioned by Tavernier.

BURMA.—*Pegu* (Prome, etc.), *Tavoy*, *Henzai* and *Tenasserim*. In Upper Burma gold is said to be plentiful at *Bhamo*, in the Hukong valley, and in the *Shan States*.

In HYDRABAD (DECCAN) are several localities where gold is found both in reef- and alluvial-grounds. Special endeavors are now being made to open up mines, but so far without great success.

MADRAS PRESIDENCY.—*Bellary*.—Gold is reported to have been found near Jagercully hill in Godda, and in *Coorg* and *Wynad*. The history of the Wynad gold-fields is interesting, although from a commercial point of view the district has been a great failure. Gold can be obtained by washing the gravels of almost all the hill-streams of Coorg and Wynad; and in the latter there are remains of ancient mines, having deep shafts through hard rocks. Notwithstanding these fair indications, expert European prospectors have failed to locate a single gold-bearing vein. Millions of dollars were spent on the strength of good reports, but to no purpose. Despite the failures that have occurred, Wynad may yet become an important gold-producing field. During the time of the Wynad boom, the Kolar gold-fields also were being investigated.

*Graphite*.—Although graphite has been known in India for centuries, the use of the domestic product has heretofore been limited to the preparation of medicinal compounds and to the manufacture of special grades of pottery. During the year 1902, however, operations were started at three graphite mines in the State of Travancore, and several thousand tons of mineral were produced. The graphite-deposit closely resembles both in quality and mode of occurrence the famous veins in the neighboring island of Ceylon, which for many years have furnished a large part of the world's output of the higher grade of crystalline graphite.

Graphite-deposits exist in many other parts of India, but better facilities for transportation are needed before they can be profitably worked in the different centers. There is a peculiar formation of this mineral in the Madras Presidency, covering an area of more than 100,000 sq. miles, which is very similar to the deposits of crystalline graphite in the United States, consisting of flakes disseminated through country-rocks, whether they be schist, granite, quartz, etc. The deposit is of poor grade, containing from 10 to 20 per cent. of graphite, but, with a suitable mechanical separator, India with her cheap labor would soon be able to produce flake-graphite of superior

quality and in large quantity to supply the ever-growing demand for this valuable mineral.

The principal localities in which graphite has been found are: TRAVANCORE, *Tinnevely*, MADRAS, *Krishna*, *Godavari*, and *Vizagapatam*; BENGAL, *Burdwan*, Raniganj and *Lohardaga* (Hutar); in CENTRAL PROVINCE, Daramgara (lat.  $20^{\circ} 24' 30''$ , long.  $83^{\circ} 18' 30''$ ), Kalahandi State, *Gurgaon*, Domaipalli (lat.  $20^{\circ} 49'$ , long.  $83^{\circ} 4'$ ), in *Kumaun* (quality fairly good, yielding by analysis from 55 to 71 per cent. of carbon); in RAJPUTANA, *Darjiling* and *Tenasserim*.

*Gypsum*.—In RAJPUTANA, CENTRAL PROVINCE, and MADRAS PRESIDENCY, the estimated production of gypsum in 1900 was valued at \$6,000.

*Iron*.—India has long been renowned for the excellent quality of its iron-ore; and in the past, large quantities were produced for domestic use, as well as for export. Although deposits of hematite, limonite and magnetite of good quality occur in many parts of India, iron-smelting is not carried on vigorously. Iron was formerly manufactured throughout India by a caste known as "Lohar," but the total production compared with the output of England was very small and expensive. In recent years the Indian demand for iron has been supplied from foreign countries, and as a result of this changed condition, the iron made by individual producers is restricted in various centers. The output is very costly, and it is being gradually replaced by cheaper imported iron and steel.

It may reasonably be asked why modern methods have not been, or should not be, introduced in India. In answer to this, India has a pitiful tale to relate. Various attempts have been made in Bengal, Madras and Central Provinces; but with the exception of Bengal, none has come to a success. The Barrakhur mines and smelting furnaces in Bengal, after years of reverses, have only of late been able to declare a dividend. The mines are very favorably situated; and as coal and flux are in close proximity, they have an advantage over the excellent Madras magnetic ore at Salem where there is no large forest or coal-deposit within several hundred miles, the nearest being the Singarini coal-fields.

CENTRAL PROVINCES promise to become a very large iron-producing center in the near future. The enormous specular-iron

deposits of *Chanda* are now being developed by an Indian syndicate of capitalists; and the India Government has established a large iron-plant in Jubbulpore, for whose requirements excellent iron-ores are near at hand. At *Bundelkhand*, *Gwalior* and many other localities, are excellent deposits of iron-ore.

The production of iron in all India in 1901 amounted to 63,000 tons, of which Barrakhur produced 57,000 tons. This quantity is only a very small portion of the supply for India's requirements.

*Lead*.—Next in importance to iron- and gold-ores is lead-ore, which was formerly mined and refined by the ancient Hindus for the lead and silver contents. The ore is generally found in metamorphic and sub-metamorphic rocks in the lower Vindyan series, but in Extra-Peninsular India and Beluchistan, it is found in Cretaceous rocks. In ancient times lead and silver were produced in RAJPUTANA, BENGAL, NWP (now UNITED PROVINCES), BURMA, AFGANISTAN and BELUCHISTAN. In UPPER BURMA, *Shan States*, are very rich lead-mines.

*Magnesite*.—The principal deposits of magnesite are in the MADRAS PRESIDENCY, at *Salem*, *Trichinopoly*, *Coimbatore* and in MYSORE. The prospects for development of the magnesite industry are excellent.

*Manganese*.—The existence of manganese-deposits in India have long been known, but the use of the ores by the ancients was very limited. The deposits are mostly in the old crystalline rocks, although in some instances they are found in the later sedimentary formations, and not infrequently in that peculiar formation known in India as laterite. The mining of this important ore commenced about twelve years ago, the main sources of supply being in MADRAS PRESIDENCY, *Vizagapatam*; CENTRAL PROVINCE; and BOMBAY PRESIDENCY. There are several other localities in Peninsular India and BURMA where manganese minerals occur, but these latter deposits have not been worked.

The manganese industry of India has increased very rapidly within the past few years, and though the local use of the ore is very limited, the quantity exported to Europe is an important factor in the world's supply. Its possibilities in the near future are very great.

*Mica*.—Deposits of high-grade mica occur in MADRAS PRESI-

DENCY at *Nellore*. Next in quality, but otherwise more important, are the deposits in CHOTA-NAGPUR, at *Hazaribagh*. Other deposits exist in several parts of India, but the production is restricted on account of the limitation of its use to the local consumption. The production and the export trade in mica, however, are increasing. The total output of mica in India during 1902 was 1,021 tons, valued at 13,13,909 rupees; of this quantity Bengal produced 833 tons.

*Molybdenum*.—Stray specimens of molybdenite have occasionally been found and collected in the museums, but so far, the localities from which they were obtained, and the manner of their occurrence, have not been ascertained.

*Nickel*.—Nickel minerals have been found solely in association with cobalt-ores, but at the present time no nickel is produced in India.

*Petroleum*.—Of the four distinct oil-fields in India, viz.:—BURMA, ASSAM, PUNJAB and BELUCHISTAN, Burma ranks first. The oil-wells in this part of the country have been worked for 2,000 years or more.

In BURMA, *Arakan*, including all the islands having oil-deposits, the prospects are very promising. In LOWER BURMA, at *Pagu*, *Prome* and *Thayetmyo*, are a number of fine deposits.

In UPPER BURMA the area under operation is very large, and the oil industry is extending year after year. The continuance of the production of these fields is difficult to calculate, for the reason that their exploitation has been going on for nearly 2,000 years, and as a result, the surface-collection has been largely exhausted. Deep-sinking will no doubt give returns for many years yet to come. The quality of Burma oil is good, but the process of refining as carried on locally can be greatly improved. The production of the Burma oil-fields in 1892 was 474,787 gal., valued at \$120,000, which was increased to 49,441,734 gal., valued at \$677,170 in 1901.

Next in importance to BURMA is ASSAM, which produced 630,000 gal. of oil in 1901. The oil-regions are in the neighborhood of the Makhun coal-field; and the occurrence of oil is intimately connected with the presence of coal-bearing rock. Marsh-gas commonly occurs. At *Cachar*, on the Barak river, near Sialtkah, are indications of a large deposit of petroleum, and it is quite within the realm of probability that Assam and

Cachar will some day rival the neighboring field of Burma. A small quantity is produced in Beluchistan, but the fields are capable of great development. The output of Beluchistan in 1894 amounted to 50,000 gal.; since then, however, the work has been suspended.

In PANJAB 2,206 gal. were produced in 1892 and only 1,812 gal. in 1901. It is evident that operations in Panjab are very slow indeed; the field has not been thoroughly prospected and its capacity, though promising great development, has not been ascertained.

*Peat.*—Extensive beds of peat are found and worked in CASHMERE; at *Nilgiri*; UNITED PROVINCES; NEPAL; ASSAM and BURMA. Attempts have been made to utilize the Nilgiri deposit for several industrial purposes in the hill-station of Otakamund. Southern India is still suffering much from the lack of coal-mines to develop mining and other industries. This deposit of peat, however, will partially supply the fuel-demand of the hill-regions.

The peat-deposit is situated at about 6,000 ft. altitude, in a manner that goes to prove the limitation of the occurrence of peat to temperate and cold regions. An elevation of 6,000 ft. virtually brings the country into a condition comparable with the temperate zone.

Peat is found in LOWER BENGAL (poor and insufficient), OUDH CASHMERE (remains of Aquatic plants and not mosses. Analysis—carbon, 37.5 per cent.), UNITED PROVINCES, NEPAL (Impure Beds), ASSAM, at Fenchgung (combustible), and BURMA (considerable deposits are met with in the higher valleys of the Irrawaddy and Salwin rivers).

*Platinum.*—Several reports of the occurrence of platinum in India and Burma have been made, but in no instance has the pure metal been found. It is always in combination with other rare metals, as iridium, osmium, rhodium and palladium. Platinum has been found in MYSORE; in Bengal, at *Singbhoom* and *Midnapur*; in BOMBAY, at *Dharwar*; in ASSAM, at Noa Dehing; and in BURMA, at *Henzai* and *Ava*. Major Baden-Powell<sup>12</sup> also mentions platinum as having been obtained by washing dirt at Safed-Cona in the Indus valley. Of the above-named

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<sup>12</sup> *Punjab Products*, p. 14.



localities, the stream-sands of Burma seem to be the most productive.

Major H. Burney describes<sup>13</sup> the peculiar method employed by the Burmans to collect gold and platinum. "The horns of a species of wild cow called "Tsain" (*Bos Sondaicus*), which are covered with a velvet coat up to the age of two or three years, are placed in the streams, and at the close of the rainy season when the water subsides, they, together with the sand surrounding them, are carefully raised with cloths wrapped around them. The velvet growth on the horns causes a concentration of gold and platinum dust."

*Salt*.—Owing to seasonal causes, the production of salt in India varies considerably from year to year. The supply is obtained from three sources, namely:—salt-lakes and wells in RAJPUTANA, rocksalt mines of PANJAB, at *Salt Range*, and sea-water near BOMBAY, SIND, MADRAS, LOWER BURMA and ADEN. The quantity produced from sea-water exceeds two-thirds of the total output. A large quantity of salt is imported from Liverpool, Hamburg, Aden, Egyptian and Red Sea ports to Bengal and Burma.

*Saltpeter*.—Crude saltpeter is obtained principally in the BEHAR from which it is sent to Calcutta for refining and subsequent export. The reported annual production averages about 21,000 tons, a quantity which is considerably below the actual output, for the reason that the annual exports in recent years have exceeded 35,000 tons. There has been a decline in the production owing to the decreased demand for use in the manufacture of gunpowder and in the preservation of food products.

*Silver*.—There are no mines worked especially for silver, nor is there any evidence that the ancients ever treated silver-ores; the silver produced by them being obtained as a by-product in the smelting of lead-ore. Silver is extracted at KANDAHAR, CASHMERE and SHAN STATES in UPPER BURMA. The last-named country, however, is the only place where the industry is regularly carried on. Colonel Yule mentions the occurrence of silver-ore on the southern crest of the Hindu-Kush mountains.<sup>14</sup> India is now one of the largest buyers of silver in the world

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<sup>13</sup> Quoted in Ball's *Manual of the Geology of India*.

<sup>14</sup> *Marco Polo*, Colonel Yule, vol. ii., pp. 325 to 327.

and, therefore, the discovery of a large deposit of silver-ore would be of inestimable benefit to the country.

*Steatite*.—This mineral is found in abundance in metamorphic rocks in several parts of India, notably in the MADRAS PRESIDENCY, BENGAL, CHOTA-NAGPUR and ORISSA. It is used in building temples and houses and to make statues of gods, etc.; but its great demand at the present time is for the manufacture of various kinds of plates, cups, saucers, and other household utensils used both by the rich and the poor. Millions of dollars' worth of steatite articles are produced every year. The famous temples of Bhubaneswar in Orissa are made of a black variety of steatite, obtained from the quarries of Neilgiri in Orissa.

*Stone*.—There are a number of varieties of building-stone in India, some of which being extremely pretty and useful. Nearly all of the material used in the building of the celebrated Taj Mehal in Agra was procured locally, as well as that of the famous modern Jain Temple in Mount Abu. The marble and limestone-deposits are of special note in the Cuddapah and Bejawar series, also in the Aravali series where the limestone-formations attain considerable size. In the Lower Vindyan series the marbles are of peculiar tenacity. The Aravali series supplied most of the material of the temples in Mount Abu.

*Sulphur*.—The main source of sulphur in India is from the deposits near Hot Springs. Next in importance is the quantity obtained from the vicinity of active or extinct volcanoes; these latter deposits, however, are almost all situated in inaccessible places, consequently the sulphur-industry in India has never been a success. With energy and capital some of these places now inaccessible may be converted into busy centers of industry. For instance, in PANJAB, Ginjauly Hills in Kohat (lat.  $39^{\circ} 29' 30''$ , long.  $71^{\circ} 48'$ ) are full of pyrites and alum-shales. There are other localities where sulphur exists, viz.:—BELUCHISTAN, *Godaveri* (Sunnee); AFGANISTAN, *Bannu*; CASHMERE, *Simla*, *Kumaun*; UNITED PROVINCE; *Jaunsar*; *Nepal*; UPPER BURMA and BARREN ISLAND, situated in the middle of the Bay of Bengal. Attempts have been made to work sulphur here, but without encouraging results. *Nepal* and the *Shan States*, UPPER BURMA, have an unlimited supply of pyrites which is used in the manufacturing of sulphur. The pyrite is found in Tertiary blue clays, found at from 12 to 20 ft. below the surface.

According to Capt. Herbert,<sup>15</sup> sulphur occurs in considerable quantities in the lead-mines at Meywer on the Tons river.

*Tin*.—Although no deposit of tin-ore has yet been discovered in Peninsular India, stray samples have been found in BENGAL, CHOTA-NAGPUR, BOMBAY and BASTAR. The only place in British India where this mineral is known to exist in a large quantity is in LOWER BURMA, at *Amherst*, *Tavoy* and *Mergue*. Tin smelting is carried on by collecting and treating tin-sand solely, although rich lodes are supposed to exist in close proximity. The matrix of tin in this part of the country is granite, and generally, wherever there is a junction of granite with metamorphic rocks, the ore is found to be abundant. Near Giridi in *Hazaribagh* is a vein of tin-ore. The Koles of this locality formerly worked the mine and smelted the ore, but at present the mines are abandoned.

*Zinc*.—Samples of blendes and zinc carbonates (smithsonite, etc.) have been found in some lead-mines, but not as a distinct body. The only mine on record is in RAJPUTANA, *Udaipur State*, at Zavar. The ore probably occurred in quartzite formations of the Aravali series. The Zavar mines have been closed since the year 1812.

#### LABOR.

In all mining-operations the question of labor-supply is the most important. Fortunately for India, labor for all purposes is plentiful and cheap. The social conditions in India, especially with regard to the artisan class, has for ages been modeled in such a way that labor, owing to the caste system, descends from father to son as hereditary; for instance, a carpenter's son as a rule would be a carpenter, a stone-mason's son would elect to be a stone-mason, etc. Since the occupation and administration of India under British rule, education with western ideas has greatly removed such conservative ideas of parents in training their children in their own profession; but this education, broadly speaking, is yet in its infancy.

The population of India, by the latest Census report (1901), is nearly 300,000,000, of whom the majority are engaged in agricultural pursuits. At present, the mineral fields of India are in southern India, Deccan and Madras Presidency, Central

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<sup>15</sup> *Asiatic Researches*, vol. xviii. (1833), p. 229.

TABLE III.—*A Summary of the Mineral Production of India During the Years 1900 and 1901.*<sup>(a)</sup>

	1900.		1901.	
	Quantity.	Value.	Quantity.	Value.
	Metric Tons.	Rupees.(b)	Metric Tons.	Rupees.(b)
Clay .....	{ 3,143,639	4,126,660	.....	.....
	{ 508	(c)	.....	.....
Coal .....	6,216,882	20,146,222	6,674,214	19,850,582
Copper-ore .....	77	244	.....	.....
Corundum .....	70	3,382	.....	.....
Diamonds .....	{ carats 169	} 11,372	.....	.....
	{ grams 34		.....	.....
Garnets .....	163.5	50,000	.....	.....
Gold .....	{ oz. 513,266 } { kg. 15,964 }	28,377,063	{ oz. 531,766 } { kg. 16,540 }	28,961,061
Granite .....	{ 367,287	288,081	.....	.....
	{ 1,472	(c)	.....	.....
Graphite .....	1,858	(c)	2,530	(c)
Gravel and rubble .....	170,075	95,818	.....	.....
Gypsum .....	4,415	1,698	.....	.....
Iron-ore .....	{ 64,085	165,924	.....	.....
	{ (c)	1,643	.....	.....
Jade .....	90	163,396	.....	.....
Laterite .....	565,231	589,035	.....	.....
Limestone .....	1,201,875	760,408	.....	.....
Magnesite .....	229	(c)	.....	.....
Manganese-ore .....	(d) 132,767	1,700,139	(d) 135,307	1,437,858
Mica .....	931	(d) 885,669	1,156	995,892
Petroleum .....	liters 171,421,086	2,231,325	liters 227,514,191	3,065,131
Rubies .....	(c)	1,459,898	.....	.....
Salt .....	1,021,426	4,585,454	1,120,239	5,622,728
Salt peter .....	11,709	1,629,657	12,751	(c)
	395,425	430,649	.....	.....
Sandstone .....	{ cu. m. 5,118	} (c)	.....	.....
	{ 7,341		.....	.....
Slate .....	{ 35	27,674	.....	.....
	{ 113	(c)	.....	.....
Soapsand .....	2,339	230	.....	.....
Soapstone .....	{ 83	30,029	.....	.....
	{ (c)	(c)	.....	.....
Stone (miscellaneous) .....	{ 269,157	81	.....	.....
	{ (c)	206,972	.....	.....
Tin-ore .....	106	5,267	.....	.....
Trap .....	756,291	128,000	71	116,595
Tourmaline .....	kg. 33	143,951	.....	.....
		18,600	.....	.....

<sup>(a)</sup> *Mines and Quarries, General Report and Statistics for 1901, Part IV.*<sup>(b)</sup> The value of a rupee in U. S. currency is \$0.324.<sup>(c)</sup> Not stated.<sup>(d)</sup> Estimated.

Province, Bombay Presidency (Dharwar), Rajputana (Khetri, Jeypore), Cashmere (Jammu), Bundelkhand, Bengal, Chota-Nagpur, Assam and Burma. In all these places, labor for mining-purposes is available. If, however, any one of a particular

class of artisans be found wanting in qualifications or in number, the importation of similar labor from a neighboring province is always adopted. For instance, in the Kolar gold-field, southern India, almost all of the labor is obtained locally, although a goodly number of miners and masons are drawn from adjoining States.

The laboring classes of southern India are not as strong a race as those from the north. In craftsmanship, all the races of India are not equally the same; the southern people may excel in one branch of art, and the northern may excel in another; for instance, the Cuchis, Guzraties, and Sindies of western India are experts in stone-masonry, the Punjabies in carpentry, the middle Hindustanies in metal-work, etc.; so that to secure a body of expert artisans one may have to get them from different places. But with the present railway-communications of India, material and labor from one district can be transported to another without inconvenience or much expense.

Usually all the imported laborers are accommodated in houses built for them, called "Lines," which are situated on high open ground, and they are under the care and superintendence of a physician. The wages of the artisans being higher than the ordinary laborers, they are sometimes allowed to live in a neighboring village. A common surface-laborer in a mine in southern India would receive on an average from 4 to 8 c. per day, in Punjab from 10 to 16 c., in Bengal (Chota-Nagpur) from 4 to 8 c., in Assam from 12 to 16 c. and in Burma from 16 to 20 cents. Of the miners, or coolies working underground under the charge of a competent miner, the aboriginal tribes of India are undoubtedly the best. They are hardy, laborious and honest in their work. In the coal-mines of Chota-Nagpur and Bengal, or in the mica-, limestone- or iron-mines, mostly all the labor is recruited from the aboriginals,—Santhals, Koles, etc. The system of underground mining is carried on either by day-labor, or by contract given out as piece-work. Very often a whole family will be found engaged in working underground, their combined labor helping to keep them out of poverty.

The pay of the artisan class, however, is better, averaging from 16 to 32 c. per day in India, and from 32 to 64 c. in Assam and Burma according to locality. Of those who are employed to look after machines and boilers, almost all are tech-

nically-trained Indians, usually under the superintendence of a competent European mechanical engineer. The pay of these men is reckoned by the month, and ranges between 10 and 30 dollars.

The demand for artisans and mechanics is on the increase, but the industrial development of India being of slow growth their condition will remain for a long while much the same as it is now. Another great factor conducive to healthy mining-operations is the absence of labor unions and labor strikes. So far, India is quite free from such labor troubles. In fact, the country cannot afford to have strikes to hamper or destroy its infant industries.

#### MEANS OF COMMUNICATION.

Under this heading are included railroads, canals, rivers or water-ways, cart-roads, and foot-paths.

*Railways.*—Looking at the map of India one can easily ascertain that the most important districts and provinces of India are now equipped with a very complete railroad-system. Commencing from Tuticorin in the south, the railroad connects uninterruptedly the Presidency of Madras with the Provinces of Bengal, up to Darjiling in the Himalayas. From Tuticorin, another line traverses the State of Mysore, Bombay Presidency, Rajputana, Punjab, up to Peshwar on the west bank of the Indus river. Great lines of railroads connect Bombay with Calcutta and Madras; Calcutta with Peshwar; and Lahore with Kurachee. The Assam-Bengal railroad, connecting Calcutta with northeastern Assam, will soon extend eastward to join the railroad-systems in upper and lower Burma. Extensive railroad-systems of nearly thirty thousand miles in India and Burma very nearly supply the facilities required for developing the mineral resources in different parts of India. The Kolar gold-field has its own railroad, and the adjacent Wynad gold-field also has a railroad. Travancore, which promises in the near future to be a busy scene of graphite-mining, is constructing a line to join the existing railroads in the north and southeast. The Dharwar gold-fields and the important manganese-beds, in the Bombay Presidency, have railroads passing through them. But in my estimation the most important railway line is the Bengal-Nagpur railroad and the G. I. P. connecting Bombay and Calcutta in a direct line, tapping the most

mineralized belt of country in Chota-Nagpur and Central Provinces. This tract of country has enormous possibilities in its mineral production. The districts of Manbhoom and Singbhoom, on this line of communication, have rich deposits of gold, lead and silver, copper and iron. In addition to rich deposits of coal in the limestone-formations, and precious stones of the corundum family; platinum and diamond also are known to exist which can probably be worked with profit. From the district of Singbhoom proceeding west, the railroad passes through the districts of Shambhalpur, Raipur, Bilaspur, Nagpur, Bhoswal, etc., across the western Ghauts to Bombay. Throughout this great length of country the laborers, especially for underground work, are excellent, for the reason that they are recruited from the aboriginal tribes of Gonds, Bheels, Koles, Santhals, Uraos, Mundas, etc. The coal-beds and fine iron-ore deposits in Central Provinces are only partially worked. The Umaria coal-mines near Jubbulpore, and the Warora coal-mines in the head-waters of the Godavari river have of late years been the source of an economic supply to the G. I. P. railroads and other industrial concerns, such as cotton-mills, etc. Its greatest possibilities exist, however, in the development of the iron-deposits and the subsequent manufacture of steel and iron; for the reason that coal and limestone are found in the same neighborhood and can be carried cheaply by rail to the furnaces.

The coal-mines of Bengal and Behar, including the districts of Burdwan (Raniganj), Santhal Purganas, Hazaribagh, Gya and Daltongung, have now a net-work of railroads for the distribution of their enormous output of coal. Although the existing lines of railroads in Chota-Nagpur, etc., serve their purpose, yet, thousands of square miles of country, known to be highly mineralized, remain unopened through lack of good wagon-roads or railroads. The Province of Tenasserim, in lower Burma, also should have railway-communications for the development of its extremely rich tin-deposits.

Additional railroad lines have been extended through many mining-fields, whose mineral wealth still awaits the enterprise and attention of capitalists for their development.

*Waterways.*—In regard to waterways, India is very favorably situated. Its large river-systems help in a great measure

internal trade and commerce. In fact, for many centuries they were the only great highways of communication and means of distributing grain and other food-products from one part of India to another. The rivers Indus, Ganges, Brahmaputra, in northern India; Irrawaddy and Salwin, in Burma; Mahanadi, Godavari, Krisna, Kaveri, Narbuda and Tapti, in southern India; are still the means of transporting goods for internal trade.

*Canals.*—In ancient India canals were constructed to connect rivers, but the system attained its present magnitude and perfection only within recent years under British rule. Many of these canals pass through mineral-districts and are of service to mining companies.

*Roads.*—Throughout India a number of grand trunk roads, as they are called, connect almost all of the important towns; but as there are yet vast tracts of jungle land and reserved forests where minerals have been located, a further extension of roads has been found necessary; this is especially urgent in the Central Provinces, Eastern Ghauts, the Agency parts of Chota-Nagpur, and Lower Burma. Footpaths and tracks for ponies and other beasts of burden are utilized as a means of transportation throughout the entire country, including the jungles.

Another great consideration for making a mining-field successful is the facility possessed by India in its supply of economic motive power for various purposes of mining and metallurgy.

*Water-Power.*—Although India possesses a great number of waterfalls, unfortunately they have not yet been much utilized directly or indirectly to drive mills for various industrial purposes; but, to the credit of that well-governed Indian Principality of Mysore, and to the healthy spirit of mining-operations which exists there, the Mysore government now possesses a powerful plant for generating electricity. The Kaveri Falls have been utilized to drive large dynamos, similar to those used at Niagara Falls, N. Y. The electric power so generated at the head-waters of the Kaveri is being conducted nearly 90 miles to the Kolar gold-fields, where it is used for mining purposes. The Kaveri Falls is not inferior in its capacity to Niagara Falls; hence, if further mining-camps be added in My-



sore, there will hardly be any difficulty concerning the future supply of power. This new power-system has indeed been a boon to the Kolar mining-concerns; because the rainfall in that part of the country is inadequate, and consequently the water-reservoirs are incapable of furnishing all the water needed for boiler- and other purposes throughout the year. Besides, every bit of coal or firewood has to be brought from a long distance, which makes steam-raising a matter of great expense. The new electric-power supply has been found to be of moderate cost.

From the head-waters of almost all of the great rivers in India inexhaustible power can be obtained. The government of India has for some time been experimenting to get the power required for Jubbulpore military works from the Falls of the Narbuda.

All who have had the experience of traveling around the hill-stations of the Himalayas or other hill-ranges know the existence of a number of water-falls, whose power may be easily utilized to supply electricity at a moderate cost. For example, the Victoria Falls at Darjiling, many falls on the road to Cashmere from Murree, and in Chota-Nagpur. The use of water-falls to supply power for the generation of electricity is best appreciated in places where coal is scarce or expensive.

*Coal-Power.*—Bengal, Chota-Nagpur, Central Provinces and Hyderabad (Deccan) are most favorably situated with regard to the generation of steam-power. Of these, on account of their coal-mines, Bengal and Chota-Nagpur in particular possess the greatest facility for making steam at a small cost. The cost of labor in Burma is high, and coal-mining in Upper Burma is found expensive. Hence Burma is supplied with coal from Bengal. However, its many rivers may be utilized to advantage for the generation of power for industrial and mining purposes.

### *Summary.*

Having noted briefly the mineral-locations, their quality, quantity and prospects, the condition of labor, the means of communication and the facility for obtaining power, it remains for me to ask the interested public, especially those who have means and capital at their command, to refer to the voluminous and trustworthy literature of the Geological Survey Depart-

ment of India, for fuller and more complete information concerning any particular mineral. Of the many ores, gold, lead, copper, iron, antimony and graphite have a great future before them. The prospect of diamond-mining may also be recommended with confidence. And I feel confident, if enterprising capitalists be discreet in securing the services of engineers and mineralogists of ability and character, success is bound to follow the mining of any of the above-named minerals. Chota-Nagpur and Burma should have the greatest claims to investigation, for the fact that they have only been partially opened and surveyed.

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### Biographical Notice of Arthur L. Collins.

BY BENJAMIN B. LAWRENCE, NEW YORK CITY.

(Albany Meeting, February, 1908.)

ARTHUR LAUNCELOT COLLINS was born at Truro in Cornwall, England, July 8, 1868. His father, J. H. Collins, is well known as an engineer, and his books have been widely read in this country. Arthur was the third son. Three of his brothers are engaged, as is his father, in the profession of mining engineering.

His education, gained partly at St. Austell, partly at Falmouth, and partly at Nuttall's Collegiate School in Truro, England, was supervised by his father, who was then Lecturer on the Mining Sciences to the Miners' Association of Cornwall and Devon.

In 1881, J. H. Collins went to Rio Tinto, in Spain, as chief chemist and metallurgist, and it was there that Arthur began his practical work in the laboratory and continued his studies in chemistry and mineralogy. To the latter science he was deeply attached; and he never lost an opportunity of adding to his cabinet of minerals, of which he was an ardent collector.

In 1883, he became assayer and general assistant to Mr. Nicol, Manager of the Peña del Hierro copper-mines, near Rio Tinto. Until 1885 he was associated with his father in the study of the geology of the Rio Tinto belt. In 1885 he returned

to Cornwall, to take charge of smelting-works near Breage; and in the same year he accompanied his father on a mining reconnaissance in Lower Burma. Upon his return he was engaged in mining-operations in both Spain and Cornwall. In 1886 he accepted the charge of the important Berkeland zinc-mines, near Stavanger, in Norway. It was here that he erected his first concentrating-mill.

In 1890 Mr. Collins visited New South Wales, Queensland, Tasmania and other British Australasian colonies, examining deposits of gold, silver, lead, tin, quicksilver and opal. In later years, he often acknowledged the valuable experiences gained during this journey.

Early in 1892 he was engaged by the late Abdurrahman, Amir of Afghanistan, as consulting geologist and engineer. This engagement, which lasted about eighteen months, was full of interest and excitement. The time was about equally divided between Kabul, the capital, Paghman, the Amir's summer residence, and field-explorations in remote parts of the Hindu Kush. His most interesting and important professional work was the development of the Amir's mines of lapis lazuli, which proved commercially valuable. During his attendance upon the court, his chief amusement (more or less obligatory) was playing chess with the Amir, whom he soon learned it was not safe to checkmate, since defeat by "a dog of a Christian" was contrary to that potentate's conception of his own dignity and sovereign prerogative! During this period, Mr. Collins became very intimately acquainted with the Amir and his sons, and with the interior workings of a despotic Oriental court.

In 1893, in consequence of the strained relations (subsequently readjusted by the "Durand mission") between the governments of India and of Afghanistan, he resigned this position, and, somewhat impaired in health by malaria, left Afghanistan.

The period immediately following was spent in professional journeys to Norway, Spain, France, Asia and Australia, which (as might be safely inferred from his personal quickness of apprehension and passion for new knowledge) greatly added to his comprehensive grasp of professional problems.

Early in 1895, Mr. Collins established himself in Gilpin

county, Colorado, where he represented English clients. One of his first steps, in this new stage of his professional career, was to ally himself with this Institute, of which he was elected a member in June, 1895, and to the *Transactions* of which (xxvi., 1041) he subsequently contributed an interesting communication on the recovery of amalgam from the old copper plates of stamp-mills.

He became, in 1895, manager of the California mine and the Hidden Treasure mill, at Central City, Colo. The old California shaft, 2,200 ft. deep, was repaired, and the mine was thoroughly equipped under his skilful direction. In the mill, he introduced valuable improvements in the handling of tailings, such as the screen-frame attachment to the Gilpin county "bumping-table," and many other devices for the increased saving of gold, so that, under his management, the Hidden Treasure mill paid substantial dividends, and is still regarded as the most satisfactory of all the mills in Central City which depend upon custom-work. This is but one instance of the way in which wide experience and the capacity for intelligent and exhaustive experiment placed him in the front rank as a practical expert. His mechanical genius was of the first order, as was shown in his new design for the construction of the frame of a stamp-mill, and afterwards in the construction of the mills of the Smuggler-Union mine at Telluride. In view of the fact that, from 1895 to 1899, Mr. Collins was connected as manager or consulting engineer, with many leading Colorado mines and mills, including the Gold Coin, Perigo, Gregory-Bobtail, Freeland, Terrible, etc., it is fair to say that he revolutionized the stamp-milling practice of Colorado by his activity in the introduction of the many improvements which had grown up slowly in Gilpin county.

In Central City, he built a concentrating-mill, which he successfully ran as a custom-mill, for the treatment of such ores as were not suitable for straight stamp-milling practice. This mill was equipped with many new appliances; and in it he conducted many experiments which were of great value in his subsequent work.

It was his intention to publish an exhaustive treatise upon stamp-milling and concentration; and it is much to be regretted that he did not live to see this purpose realized. His mechan-

ical bent urged him very strongly in the direction of mill-work; and his deep interest in this line has been very widely appreciated by those who from time to time have read his articles upon this subject in the scientific journals. His activity also found an outlet in taking leases upon abandoned mines in the Gilpin county region; and with his two brothers, Edgar and George, he became interested as lessee in a number of the mines. Among the most successful of his efforts was the re-opening and equipment with a 30-stamp mill of the Perigo mine in Gilpin county. He considered this mill one of his best achievements.

His experience in Central City, covering a period of five years, fitted him for the position which was offered him as manager of the Smuggler-Union mines at Telluride, Colorado. These mines produced a very complex ore, requiring the greatest ingenuity and care in handling. Under his direction a 60-stamp mill was built and equipped, which is to-day a witness to his peculiar experience and genius in the milling of gold-silver ores.

The closing years of his life were signalized by a struggle with the Telluride miners' union, then dominated by an anarchistic spirit, which terrorized the whole district. His courage and devotion under these conditions gained for him the support and friendship of the best citizens throughout the State of Colorado; but they made him a marked man for the revenge of cowardly antagonists. His life was continually threatened, but he persistently refused to leave his post or surrender to the forces of disorder. Finally, on the 19th of November, 1902, while at home with friends in the Smuggler-Union office, he was shot by an unknown assassin; and twenty-eight hours afterwards he expired.

## The Copper-Deposits of the Kaibab Plateau, Arizona.

BY E. P. JENNINGS, SALT LAKE CITY, UTAH.

(New York Meeting, October, 1903.)

THESE unique copper-deposits occur on the top of the Kaibab Plateau, in Coconino county, Arizona, and extend from the northern edge of the Grand Cañon of the Colorado river to near the Utah State line. The best-developed deposits are near Jacobs Lake, 30 miles south of the Utah line, where they are 16 ft. thick, of unknown width, and are more or less continuous for 5 miles.

From Jacobs Lake to the edge of the Grand Cañon, a distance of 40 miles, outcrops of ore have been found, but no explorations have been made except those of wandering cowboys and hunters.

The Kaibab Plateau, the highest of the Colorado river plateaus, is a great uplift, about 90 miles long from north to south, and from 10 to 20 miles wide from east to west. Its southern face is the Grand Cañon; its western boundary a N-S. fault plane, which separates it from the Kanab Plateau, lying 1,500 ft. below. Its eastern edge is a monocline which brings the horizontal beds of the high plateau to the level of the plateau-country of eastern Utah.

The elevation of the plateau near the Colorado river is over 8,000 ft., and about 6,000 ft. at the Utah line. Its structure, as shown by the western fault, consists of about 500 ft. of Aubrey limestones underlain by 1,000 ft. of red sandstones. The limestone is the upper member of the Carboniferous, and is overlaid by the marls and shales of the Permian, 25 miles to the north on the Kanab Plateau. At the town of Kanab, near the Utah line, the Permian strata pass under the magnificent red-sandstone cliffs of the Triassic, that rise to heights of 1,000 ft. above the desert. The Grand Cañon exposes a section a mile deep at the southern end of the plateau, showing all the formations from the Archean to the top of the Carboniferous.

This mountain-mass, cut by two sections, the cañon and the western fault, shows no mineral veins or mineralization, above the Pre-Cambrian, near the bottom of the cañon. These ancient metamorphic strata contain a few copper-bearing dikes, with the usual contact-impregnation of copper-ore and copper-stained rock. All the Paleozoic strata are barren, both in the cañon and along the western fault, until the copper-beds are reached at the top of the plateau.

I have described the formation of the plateau, for the purpose of showing that the possibilities of any connection between the copper-veins and dikes of the Pre-Cambrian, and the horizontal copper-beds of the Carboniferous limestones are, to say the least, very remote.

The ore-beds are white chert, impregnated with malachite and azurite; small amounts of earthy cuprite, copper glance and chalcopyrite also have been observed. Silver and gold are present in very minute quantities. The ores range from 2 to 40 per cent. of copper, and average 7 per cent. They are too siliceous for smelting, but are good leaching ores.

Some leaching-experiments have been made in a small way, but no copper has been produced commercially. The ore is covered by a few inches of soil, and often outcrops. The upper portions of some of the beds have been leached, leaving a mass of honey-combed iron-stained rock. The beds are intersected by many small faults; the ore near these faults has been crushed and recemented by silica and copper, showing that the period of deposition was long. The quantity of ore cannot be accurately estimated from the small amount of development work done, but it is believed that 1,000,000 tons of 7 per cent. ore can be put in sight at the Jacobs Lake beds, alone.

I am not aware that any theory has been advanced to explain the occurrence of these ores. Evidently, they are replacements of the limestone by silica and copper. The ore-bearing solution was a descending carbonated alkaline water, which dissolved both silica and copper from the Permian and Mesozoic measures, that, in former time, covered the Carboniferous. This theory is rendered more probable from the fact that the Triassic sandstones to the north and east are copper-bearing, and sufficiently so, to be mined at a profit. There are no known eruptives that could furnish the copper, except the dikes of the Pre-Cambrian,

a mile below the present copper-horizon. I consider it a case of solution from a great mass of superimposed rock, and the precipitation by chemical means at one horizon; the Aubrey limestone being the precipitant.

The deposits can be considered as horizontal bedded-veins, from which the hanging-wall or roof, and also much of the ore, has been removed by erosion. These beds are similar in physical characteristics to the iron-deposits of the Mesabi; the taconite or iron chert of the latter corresponding to the copper chert of the Kaibab. The chemistry of the formation of the two may be alike, but the Kaibab ores lack the subsequent enrichment the iron-ores have received.

Being far removed from any library of reference, I offer these incomplete notes more as a subject for discussion, than as a contribution of scientific value. The subject of bedded-veins will bear much more discussion than it has yet received.

## Notes on the Coal- and Iron-Fields of Southeastern Shansi, China.

BY WILLIAM H. SHOCKLEY, LONDON, ENGLAND.

(New York Meeting, October, 1903.)

### INTRODUCTION.

THOUGH China has been widely explored by mining engineers during the past dozen years, comparatively little has been published concerning its mineral resources. The few monographs scattered through the transactions of various societies contain far less information than the private reports made to the numerous syndicates which have sent out engineers. The best general account of the geology and mineral resources of China is to be found in the works of von Richthofen,<sup>1</sup> a remarkably gifted observer.

The observations recorded here in this paper were made in the service of the Pekin Syndicate (which has kindly given me permission to publish them), during the first months of 1898, among the coal- and iron-fields of southeastern Shansi. This is one of the largest anthracite-regions known, the area being

<sup>1</sup> *China, Ergebnisse Eigener Reisen und darauf Ge gründete Studien.* Berlin, 1877.



about 13,500 sq. miles. Von Richthofen estimates the thickness of the coal at 12 m. and the total coal at 630,000,000,000 tons. I should put the average thickness of the coal somewhat lower; but there are not enough data for certainty, and Von Richthofen's estimate is as likely to be fair as any other that can now be made.

According to Mr. E. W. Parker,<sup>2</sup> the world's production of coal of all kinds during 1902 was 884,803,434 short tons. At that rate of annual consumption, this field contains enough coal to supply the whole world for the next 700 years; or, if we estimate upon a smaller thickness, 450 years. But this includes only the anthracite-fields of southeastern Shansi; the bituminous-fields of western Shansi are nearly as large; and northern Shansi also contains large deposits of coal, which extend westward and northward into Shensi and Kansuh. It is safe to say there is enough coal stored away here for the needs of the world for more than a thousand years. There is a large quantity of coal mined annually in China, which is not included in any general statistics that I have seen; the estimate just quoted classes "China, Turkey, Servia, Portugal, Colombia, Chile, Borneo, Labuan, Peru, Greece, etc.," under the general heading, "all other countries," with a total product of 2,240,000 tons. The product of China alone is certainly many times as great as this. In the one province of Shansi, which is supposed to have a population of from fifteen to twenty millions, nearly all inhabitants burn coal; and an estimate of 5,000,000 tons for this one province would not be excessive; the coal used in the Shansi iron-furnaces alone would amount to several hundred thousand tons.

The coal is a hard, compact, very pure anthracite, burning readily with little flame and no smoke, without change of form, and leaving a small amount of white ash. It is very solid and will bear any burden that could be put on it in a blast-furnace. Its composition is shown by the analyses in Table I., made by Mr. Edward Riley, upon large samples sent to London.

The iron-industry of Shansi was the most important in China until the Viceroy Chang Chi Tung established the Han Yang works, near Hankow. At the present time, about 50,000 tons of pig-iron are made in Shansi annually, and the price at Tai

<sup>2</sup> *The Mineral Resources of the United States*, 1902, U. S. Geological Survey, 1904.

TABLE I.—*Analyses of Shansi Anthracite.*

	A.	B.	C.	D.	E.	F.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Carbon.....	84.75	88.00	85.01	84.67	89.14	89.24
Sulphur.....	0.35	0.37	0.58	0.41	0.44	0.31
Ash.....	12.04	8.82	11.61	12.20	7.68	6.91
Moisture.....	2.86	2.81	2.80	2.72	2.74	3.54
Total.....	100.00	100.00	100.00	100.00	100.00	100.00
Calories by Thompson's calorimeter.....	5,825	6,790	6,576	5,727	6,366	7,166

*Localities.*—A, Tse Chou; B, Siu Wu; C, Ping Ting Chou; D, near Ching Hua; E, near Huai Ching Fu; F, near Tai Yang.

Yang (17s. 2d. per ton) is probably the lowest in the world. Wrought-iron is sold for about £3 per ton. The Shansi cast-iron is of good quality. It is used for making various utensils, chief among which are the large cooking-pots, found everywhere in northern China, and largely exported to Manchuria.

The iron-ores are mainly limonites and hematites, occurring in Carboniferous shales and sandstones, as nodules, usually varying from a few pounds to a few hundred pounds in weight, though masses of several tons are said to have been found. Sometimes there are several layers of these nodules, sometimes there is only a single stratum. Iron-ore occurs also in beds from a few inches to a foot or so thick, and in flat veins with a maximum thickness, at present, of 3 ft., though tradition says that there have been beds of 20 ft. thickness found in the past. The native methods extract from 25 to 35 per cent. of iron from these ores. The analyses in Table II. show the composition of the ore.

TABLE II.—*Analyses of Iron-Ore from Tai Yang, near Tse Chou Fou.*

Components.	I. Per Cent.	II. Per Cent.	Components.	I. Per Cent.	II. Per Cent.
Silica.....	4.67	11.150	Sulphur.....	0.074	0.016
Peroxide of iron.....	76.97	65.000	Oxide of copper.....	nil	.....
Alumina.....	3.46	6.424	Carbonic acid.....	9.37	2.700
Oxide of manganese.....	0.57	0.512	Combined water.....	0.59	7.350
Lime.....	2.21	5.500	Moisture.....	1.61	.....
Magnesia.....	trace	0.252			
Phosphoric acid.....	0.57	1.076		100.094	99.980
Arsenic acid.....	nil	trace			
			Metallic iron.....	53.880	45.500

I. By Edward Riley; II. By Pattinson and Stead.

The following itinerary, giving dates, distances and altitudes, may be useful to later travelers in this province.

*Itinerary.*

Date, 1898.	Place.	Distance in Li (3 per mile).	Altitude in Feet.	Figures thus (2,860R) are von Richthofen.
Jan. 9.	Peking, . . . .	. . .	120	
	Ch'ang Hsing Tien, . . . .	40	All these places have about the same altitude as Peking, being in the Great Plain.	
	Tou Tien, . . . .	50		
" 10.	Sung Lin Tien, . . . .	60		
	Pei Ho, . . . .	50		
" 11.	An Su Hsien, . . . .	60	175	
	Pao Ting Fu, . . . .	50		
		310		
Jan. 12.	Ta Chi Tien, . . . .	...		
	Fang Shun Ch'iao, . . . .	60		
" 13.	Ch'ing Feng Tien, . . . .	60		
	Ming Yueh Tien, . . . .	60		
" 14.	Fu Ch'eng I., . . . .	75		
	Cheng Ting Fu, . . . .	40	Altitude as above.	
" 15.	Huai Lu, . . . .	60	400	
	Remained in Huai Lu, till—			
" 18.	Hung Kou, . . . .	50	850	Very bad road, paved or else rocky.
" 19.	He T'ao Yuan, . . . .	45	1,600	
	Ku Yi P'u, . . . .	35	2,800	
" 20.	Hsi Ch'iao P'u, . . . .	50	2,450	
	P'ing Ting Chou, . . . .	15	2,700 (2,860R)	
" 21.	Shih Pu Tsui, . . . .	25	2,600	
	Hsin Hsing T'an, . . . .	15		Slight snowstorm.
" 22.	Ch'in Ch'uan, . . . .	40	3,800 (4,440R)	Stormy day.
	Huang Men, . . . .	35	3,900	
" 23.	T'ai An I., . . . .	40	4,150 (4,325R)	
	Summit of Pass, . . . .	...	4,880 (4,860R)	
	Shih T'ieh, . . . .	35	4,250 (4,210R)	
" 24.	T'ai Yuan Fu, . . . .	85	2,870 (2,800R)	
		825		
	Remained in T'ai Yuan Fu, till—			
Feb. 2.	Hsiao Tien Chen, . . . .	30	2,900	
	Pei Ke, . . . .	30	2,900	
" 3.	Hsu Kou, . . . .	20	2,850	
" 4.	Pai Keni Chen, . . . .	40	.....	
	Tzu Hung Kou, . . . .	35	3,050	
" 5.	Lai Yuan, . . . .	40	3,850	
	Nan Kuan, . . . .	20	4,500	5,200 Feng Shui Ling (pass).
" 6.	Ch'uan Tien, . . . .	50	4,150	
" 7.	Chiao K'ou, . . . .	35	3,850	
	Chin Chou, . . . .	35	3,650	
" 8.	Hsin Tien, . . . .	35	3,650	
	Ssu T'ing, . . . .	25	3,500	

Date, 1898.	Place.	Distance in Li (3 per mile).	Altitude in Feet.	Figures thus (2,860R) are von Richthofen.
Feb.	9. Chiao Ch'uan Kou, . . .	40	4,000	
	T'un Liu Hsien, . . .	40	3,400	
"	10. Shih T'sun, . . .	30	3,250	
	Lu An Fu, . . .	30	3,540	
		535		
		Remained in Lu An Fu, till—		
"	13. Ching Fang, . . .	30	.....	
	Yin Ch'eng, . . .	30	4,060	
		In Yin Ch'eng, till—		
"	17. Yao Tzu Kou, . . .	35	3,650	Snowstorm.
"	18. Kao P'ing Hsien, . . .	35	3,300	
		In Kao P'ing Hsien, till—		
"	24. Ch'iao T'sun, . . .	30	3,100	
	San Chia P'u, . . .	10	3,150	
"	25. Ta Yang Chen, . . .	15	3,400	(3,260R)
		In Ta Yang Chen, till—		
Mar.	2. Wang T'ai T'sun, . . .	35	3,250	
"	3. Tse Chou Fu, . . .	20	2,810	
		In Tse Chou Fu, till—		
"	8. Er Shih Li P'u, . . .	20	4,200	
	Chou T'sun Chen, . . .	30	2,900	
"	9. Yang Ch'eng, . . .	40	2,600	
		In Yang Ch'eng, till—		
"	12. Summit of pass, . . .	...	3,600	
	Liu T'sun Chen, . . .	50	2,650	
"	13. Summit of pass, . . .	...	3,800	
	Ch'in Shui Hsien, . . .	50	3,470	(3,230R)
	Wang Chai Chen, . . .	25	.....	
"	14. Tung Wu Ling, . . .	...	5,200	(4,910R)
	Valley, . . .	...	4,750	
	Hsi Wu Ling, . . .	...	5,050	(4,830R)
	Lung Hua Chen, . . .	50	4,000	(3,970R)
"	15. Yi Ch'eng, . . .	40	2,200	
	Ch'u T'sun Chen, . . .	35	2,400	
"	16. Shih T'sun, . . .	40	2,100	
	P'ing Yang Fu, . . .	60	1,800	(1,555R)
		In P'ing Yang Fu, till—		
"	18. T'ien Ching T'sun, . . .	25	1,850	
	Hung Tung, . . .	35	2,150	
"	19. Yao Tzu Chen, . . .	35	.....	
	Ho Chou Ch'eng, . . .	40	2,450	
"	20. Jen Yi Chen, . . .	60	3,200	
	Han Hsing Ling, . . .	...	4,475	(4,145R)
	Ling Shih Hsien, . . .	40	2,950	(2,580R)
"	21. Liang Tu Chen, . . .	30	3,000	
	Chieh Hsiu Hsien, . . .	50	3,100	
"	22. Chang Lan Chen, . . .	40	3,100	
	P'ing Yao Hsien, . . .	40	2,950	(2,810R)
"	23. Ch'i Hsien, . . .	50	2,900	

Date, 1898.	Place.	Distance in Li (3 per mile).	Altitude in Feet.	Figures thus (2,860R) are von Richthofen.
Mar. 24.	T'ai Ku, . . . .	50	2,900	
	Hsu Kou, . . . .	30	(3,100) average of two determina- tions is 2,975 (2,260R)	
	Hsia T'ien Tzu, . . . .	50		
" 25.	T'ai Yuan Fu, . . . .	30	2,870	
	In T'ai Yuan Fu, till—			
" 30.	Ming Ch'ieh I., . . . .	50	3,400	
" 31.	Shih T'ieh, . . . .	35	4,220	
	Ch'ng P'ing Chen, . . . .	50	4,120	
April 1.	Ch'ao Yu He, . . . .	60	4,850	
	Yu Hsien, . . . .	40	3,850	
" 3.	Kuan T'ou, . . . .	50	4,350	
	P'an T'ou Ch'i (pass), . . . .	...	4,900	
	Sha K'ou, . . . .	30	2,450	
" 4.	Mao Er. Shih, . . . .	40	1,150	
	Wen T'an, . . . .	40	950	
" 5.	P'ing Shan Hsien, . . . .	40	700	
	Huai Lu, . . . .	60	See above for altitudes.	
April 6.	Cheng Ting Fu, . . . .	60		
	Fu Cheng I., . . . .	40		
" 7.	Ming Yueh Tien, . . . .	75		
	Ch'ing Feng Tien, . . . .	60		
" 8.	Fang Shun Ch'iao, . . . .	60		
	Pao Ting Fu, . . . .	60		
" 9.	An Su Hsien, . . . .	55		
	Pei Ho, . . . .	60		
" 10.	Sun Lin Tien, . . . .	60		
	Liu Li Ho, . . . .	50		
" 11.	Peking, . . . .	100		
		<u>2,460</u>		

*Summary of Distances in Li.*

Peking to Pao Ting Fu, . . . .	310
Pao Ting Fu to Huai Lu, . . . .	355
Huai Lu to T'ai Yuan Fu, . . . .	470
T'ai Yuan Fu to Tse Chou Fu, . . . .	775
T'se Chou Fu to P'ing Yang Fu, . . . .	440
P'ing Yang Fu to T'ai Yuan Fu, . . . .	605
T'ai Yuan to Huai Lu via P'ing Shan, . . . .	495
Huai Lu to Peking, . . . .	<u>680</u>

Total Li, . . . . . 4,130, equal to 1,377 miles.

NOTE.—The Chinese "Li" is counted as three per mile, but it will average less than one-third of a mile in length. The altitudes given in this itinerary were from observations made with a pocket aneroid of good make, and were corrected by comparison with a record kept by Professor Russell in Peking. We had much stormy weather on the journey, and the Peking record shows large variations in the barometer readings. I think, on the whole, the altitudes are correct, within 200 ft., and some are much closer.—W. H. S.

## GENERAL TOPOGRAPHY AND GEOLOGY.

The Great Plain, largely composed of fertile loess brought down from the mountains by the different rivers, stretches from Mi-Yun, northeast of Peking, in lat.  $40^{\circ}20'$  N., southward to lat.  $30^{\circ}$  N., a distance of about 550 miles, with an average width of about 175 miles, not including the large arm that reaches from near Khai-Feng lat.  $34^{\circ}40'$  N., following the old bed of the Yellow river and the Grand Canal to the plain of the Yang-Tse, near Shanghai.

East of this plain stands the mountain peninsula of Shantung; to the northwest, the loess and sandstone plateau of Shansi is cut off by the mountains of Chili; further south, the plateau itself forms the boundary.

The Great Plain is crowded with innumerable villages, towns and cities. No accurate statistics are available, but the population of the plain probably exceeds 50,000,000; and when proper transportation facilities are provided, this vast swarm of people will consume an immense quantity of coal and iron from the mines of Shansi. The mountains of Chili and Honan, west of the plain, contain small coal-fields, much disturbed and broken, in many places, as in the mountains west of Peking, by the intrusion of eruptive rocks. Behind these mountains lies the plateau of Shansi with its vast coal-fields.

The geological structure here is extremely simple and uniform. Its principal features are a lower limestone, several thousand feet thick (the "kohlenkalk" of von Richthofen), on which rests the coal-formation, usually of shales and sandstones, but sometimes with one or more strata of limestone, the whole series being usually only a few hundred feet thick. One seam of coal, from 10 to 40 ft. in thickness, is everywhere present; and in most cases there are several other seams, the combined thickness of all the seams sometimes reaching 60 ft. as a maximum. Above the coal-formation come the "gay-colored" (*buntgefärbten* of Richthofen) sandstones and shales, marking with varied shades of red, brown, yellow and green the landscape of the upper plateau.

The loess, which covers the hills and fills the valleys of a large portion of Shansi, forms characteristic deep ravines, several hundred feet wide with nearly vertical sides frequently more than 200 ft. high, which make railway-building difficult

and expensive. Over nearly all of Shansi the strata are almost horizontal. The limestone is usually found at the altitude of about 2,000 ft., while the sandstone reaches 6,000 ft. The persistence of the coal-seam is remarkable. Wherever erosion has gone deep enough to expose the coal-formation, it is never wanting unless the erosion has removed all the beds down to the lower limestone; and it is visible everywhere unless covered with loess or sandstone, under which it is undoubtedly persistent. In a part of Shansi, granite and gneiss mountains are found; near them there is naturally no coal.<sup>2</sup>

The geological structure of Shansi is so uniform that wherever in a stream-bed both limestone and sandstone pebbles are found, it is certain that coal-beds are near.

#### DETAILS OF LOCALITIES.

*Huai Lu.*—This place, situated on the border of Shansi, 220 miles from Peking, is practically the center of the export trade in coal and iron for a portion of Shansi, namely, the P'ing Ting Chou district, some 60 miles away. Pack-animals are used exclusively in the transportation. From statistics collected for me in 1898, I estimated that 100 tons of coal, 20 tons of iron, and 70 tons of other goods were thus brought eastward to Huai Lu daily, in something more than 3,000 separate loads. The return-freights were much smaller.

The coal sells at Huai Lu for from 5 to 6 cash per catty, or about 25s. per ton. It is shipped from there in carts to the Great Plain, and we found that it sold at Cheng Ting for 8 cash per catty, or about 31s. 4d. per ton. The iron brought to Huai Lu comprises hundreds of different articles, the most numerous being cast-iron pans, up to 40 in. in diameter, which are used universally for heating water. Hammers up to 50 lb. in weight, wheelbarrow-axles, shovels, axles for the stone rollers with which corn is ground, parts for wheels and fittings for carts, small stoves, and bars of small size, are the principal cast-iron articles kept in stock. There are not many articles made of wrought-iron; tires for cart-wheels and nails are the most common. They are all of native manufacture and coarse

<sup>2</sup> These remarks refer specially to the southern portion of Shansi, mainly south of T'ai Yuan, and no attempt is made to describe the geology of northern Shansi.—W. H. S.

workmanship. No foreign iron is sold here. I saw one cast-iron pan of foreign make, perhaps imported as a sample. It was much better made than the native pans. The prices ranged from about a penny a pound for the cast-iron bars to 24s. each for the largest pans.

Huai Lu is very prettily situated just at the beginning of the mountains which form the western boundary of the Great Plain, and a hill near the town affords a fine view of the plain, and the hundreds of villages and towns of its dense population.

*Shih Pu Tsui.*—This was the first coal-mining locality visited. The mines were not working, and could not be inspected; but we saw a quantity of coal in large lumps ready for shipment. This coal is of very good quality; an analysis made by Prof. Stuhlmann, of Peking, showed only 2.13 per cent. of ash. It is sold by the donkey-load for 100 cash, or about 3s. per ton. The seven or eight mines at this place employ about 100 men. They are opened by vertical shafts from 150 to 160 ft. deep. The coal lies flat, and the mines could easily be worked by adits; but the Chinese prefer the shafts, because the good coal can be reached more quickly. The country being flat, a long adit would be required to reach the point attained by a short shaft. The Chinese nearly everywhere prefer the sinking of a shaft to the driving of an adit more than a few hundred feet long. There are some exceptions, as at Yin Cheng, where there are adits of 3,000 ft. or more. The men here say they earn 150 cash, or about 4½d., per day.

The outcrop of the coal is very plainly seen soon after leaving P'ing Ting Chou; and there were mines on the other side of the valley from Shih Pu Tsui which we did not visit. We were told that at Ying Ying, 20 li north of Shih Pu Tsui, there were many big mines, some of them opened by tunnels large enough for donkeys. Soon after leaving Shih Pu Tsui we passed out of the coal-measures into the overlying sandstone, and saw no more coal-mines until we reached Yin Ch'eng, a fortnight later.

*Yin Ch'eng.*—This small but busy town is a center of the iron manufacture of the region. There are a number of iron-furnaces near by, and the pig- and wrought-iron made at Kao P'ing is also brought here and manufactured. Accurate sta-



tistics of the population of Chinese towns being unattainable, I estimate the number of inhabitants at 5,000, all engaged in the iron industry. They have a saying—"We eat iron." Although pig-iron is made here, only castings and wrought-iron are exported. The iron is cast in small cakes each weighing about 13 lb. (10 catties), but the unit employed in speaking of pig-iron is the p'ao of 90 lb. (67 catties).

Table III., which gives a statement of the capacity of the iron-works in the region of which Yin Ch'eng is the center, was obtained at that place, and, though probably neither exact nor complete, may give a general notion of the industry.

TABLE III.—*Capacity of Iron-Works in the District Surrounding Yin Ch'eng.*

Locality.	Distance from Yin Ch'eng.	No. of Shops.	Total Capacity P'ao Per Day.
	LI.		
Shan Hou T'sun.....	6 S.	3	24
Kao Tse T'sun.....	25 SW.	5	24
Shih Pan P'o.....	30 SW.	5	36
Ch'en Shu Ho.....	25 SW.	10	36
Hu Chuang T'sun.....	30 SE.	5	24
Pai T'u Wan.....	35 SE.	3	36
Tung Nan Chuang.....	20 SE.	8	36
Total.....	.....	39	216

This indicates a total daily capacity of 14,472 catties or about 100 tons. The Chinese thought that, on the whole, the shops might be turning out half of this, or say 50 tons per day.

Iron-mining has been extensively carried on about Yin Ch'eng. For a long distance the hillsides are covered with old dumps, showing that the iron-deposits extend over a large area. At the time of my visit, mining was not active, as many of the mines had not resumed work after the New Year's holiday. I visited two places where work was going on. At the first place they had a shaft down 80 ft. but had not yet struck the ore, the miners were hopeful and said that out of 10 shafts sunk anywhere in this district eight of them would find ore. The strata shown by this shaft were:—Surface dirt, 10 ft.; sandstone, coal, 5 ft. (this is the main seam of the district, but was fully decomposed here), under this same sandstone and below is a soft clay-shale in which the iron-ore is found. This clay-shale

is called "kang" by the Chinese. There are many layers or strata of ore recognized by the miners, who say that there are nine or ten in the district, though the complete series is not often found in one shaft. The seams all have names, and the miners say they can easily tell from what seam a hand-specimen of ore comes. As a matter of curiosity I give the names of the various ores from the surface downwards:—

Upper 5 layers, .	{	P'ai Kung,	First layer ore.
		Wu Hua Kung,	Five flower ore.
		Huang Mien Kung,	Yellow face ore.
		Pa I Kung,	Eight good friends ore.
Boundary, . .	{	Ti Kung,	Earth ore.
		Huo Shih,	Flint.
Lower 5 layers, .	{	Sha Kung,	Sand ore.
		Huang Ni Kung,	Yellow mud ore.
		Feng O Kung,	Wind goose ore.
		P'ei Kung,	?? " "
Below this, . .		<i>Water reaching to the sea; full of dragons and dangerous to disturb!</i>	

The best ore was the Wu Hua Kung, but this has been mainly worked out; that is, they have worked as far as they are able to without machinery of any kind and without powder or proper tools. All the miners unite in saying that there is plenty of ore to be had further in the mountain.

I went down another shaft, a few li SE. of Yin Ch'eng, and 25 ft. deep, with an incline-drift from the bottom, just large enough to crawl through. They had two seams of ore, the Pa I Kung and the Ti Kung, both being mixtures of earthy hematites and limonites. They do not form continuous beds but make flat-lying bunches in the clay slate. The ore I saw here was about 3 in. thick; and the miners say that in this district more than 5 in. is unusual, and 6 in. is a maximum. These miners are wretchedly poor; they do not make more than 100 cash a day, and many of these, they say, are "small cash"; thus their wages are not over 3d. per day. They live principally on yellow millet, which they eat with a little salted cabbage.

Though a good deal of business goes on at Yin Ch'eng, no one seems to make much out of it, and the town has a poverty-stricken air. The iron-trade has been steadily declining for many years, and with especial rapidity, as I was told by Mr. Stanley

Smith, of the China Inland Mission, at Lu An Fu, since about 1888, by reason of the competition of foreign iron and iron from the new works at Hankow, supplying markets where formerly nothing but Shansi iron was sold. From all I could learn, the product of iron in Shansi is to-day not more than one-third as much as it was 30 years ago.

All the Shansi iron is made in crucibles. At Yin Ch'eng these are 19 in. high, by 6.5 in. in diameter; they are filled with a mixture of ore which has been broken and sorted to small-walnut size, coal of about the same size, and "hei t'u" (black dirt) or decomposed coal from near the surface. The furnace-men say that this process has been used from remote antiquity, and that the use of the decomposed coal is an important part of the process. I judge the advantage of using this to be that the sulphur and other impurities have been removed by oxidation. The proportions at the works I visited were 4 baskets of ore, and 1 basket each of small coal and hei t'u. A view of an iron-furnace is given in Fig. 1.

The crucibles are heated in a stall-furnace, with two side-walls 36 in. high; the rear wall being formed of a small hut which shelters the men who work the windbox furnishing the blast. The floor of the furnace, 50 by 78 in. in area, is first covered with clay; then comes a layer 7 or 8 in. thick of coarse coal; on this 66 pots are placed and the spaces are all filled with small coal, and a few inches of small coal are placed on top; over all is a cover several inches thick of cinders and ashes, and the front of the furnace is built up of crucibles that have been used. It takes about 16 hours for a heat; the furnace is then allowed to cool for a short time and the crucibles are taken out with long tongs, and when cold they are broken and a cake of iron is found at the bottom if all has gone well. Wrought-iron is made from the cast-iron by heating in a small furnace with wood for fuel, and hammering while hot, with light hammers, as at Kao P'ing. The men are paid on a co-operative system; at the time of my visit they were earning about 110 cash or  $3\frac{1}{2}$ d. per day. Each iron-works has a number of furnaces, and judging by the great piles of slag and cinders the making of iron here must have been carried on for many generations.

On market-day, people came to Yin Ch'eng from the sur-

rounding towns, to sell their manufactured iron. Nails were the most prominent article for sale, but there was a considerable variety of other goods, among them being chains, staples, hinges, hooks for carrying-poles, ladles (some of them lined with copper), bells for donkeys and for camels; all these were very coarsely made, but the prices were low.

There seems to be only one seam of coal at Yin Ch'eng. This can be seen cropping out on the hillsides, and adits could be run here very easily. The coal is very soft and small where it crops out, but at a depth of from 100 to 150 ft. below the surface it becomes solid, and is 18 ft. thick. The natives have run adits, some of which are several hundred feet long, for the "hei t'u," or rotten coal, which they sell to the furnace-men for 50 cash per cart-load of 300 catties, or about 8*d.* per ton. I went down one of the mines producing solid coal. This was opened by a vertical shaft 180 ft. deep, for the first 50 ft. walled with stone and 7 ft. in diameter; lower down, in the solid rock, 4 ft. in diameter. As there was but little demand for coal at the time of my visit, only four or five men were working underground, each breaking and dragging to the shaft 20 baskets of coal per day, for which he got 200 cash (6*d.*). The drift from the bottom of the shaft was very small, and it was with great difficulty that I managed to crawl to the end of it, which was said to be 800 paces from the shaft; the bottom of the drift was very uneven as it ran up and down following the caves in the roof. No timbers were used, and the drift was kept as small as possible to avoid danger from a caving roof, which was sandstone, and did not seem very solid. I found at the end of the drift about 15 ft. of very solid clean coal of a soft lustrous black, with some patches of brilliant jet; I could not see the floor, as it was covered with several feet of coal, but I have no doubt the seam was fully 18 ft. thick. This coal is hoisted on a windlass by six men, who are paid 2.5 cash per basket, so the statement of the mine as given to me is:—

100 baskets of coal per day, selling at 30 cash each,	3,000 cash.
Less hoisting at 2.5 cash per basket, 250 cash.	
Wages of 5 miners at 200 cash,	1,000 cash.
	<u>1,250 cash.</u>
Balance,	1,750 cash.

The baskets of coal weigh about 70 catties each, so that the

coal sells at the mine for about 1s. 6d. per ton. The miner with whom I went through the mine was very anxious that I should advance him the money to open another mine; he said this would take about 1,000 taels (£150).

I could get no accurate information as to the amount of iron made at Yin Ch'eng yearly. It is probably at least 6,000 tons; the Chinese estimated 15,000. As to prices, my Chinese secretary was told that the wrought-iron was all of about the same quality, the articles varying in price in proportion to the skill of the man who made them; that the average price for cast-iron was 10 cash per catty (35s. per ton), and for wrought-iron, 20 cash per catty (70s. per ton). The official representative of the Governor, who went with us during the trip said, however, that the people would not tell the truth or name fair prices, because, as we had come directly from the Governor, they were afraid to give any information which might be used against them for purposes of taxation.

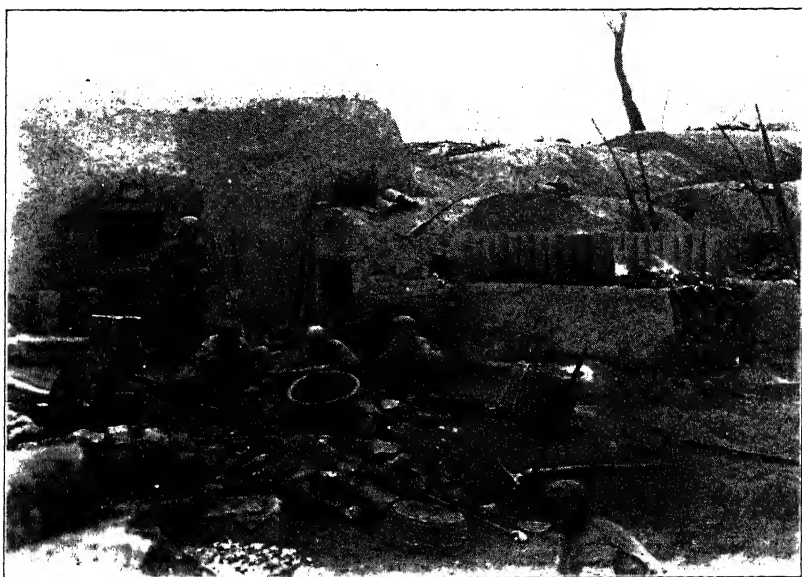
*Kao P'ing Hsien.*—In going from Ying Ch'eng to Kao P'ing Hsien, we soon entered the upper sandstone of the hills; but we were told of coal-outcrops in the valleys below, and coal-mines all along the valley-road. We met many small carts, each carrying about 500 lb. of iron to Yin Ch'eng.

The iron made at Kao P'ing is reputed the best in Shansi. The size of the cinder- and waste-dumps, some of which are visible for miles, indicates that the business has been carried on here for centuries; but here, as everywhere else, the industry is not prosperous. The ore is very similar to that of Yin Ch'eng, but the seams are thicker, and the ore is found over a very large extent of country.

The first mine visited, 5 li west of town, was 50 ft. deep, and had an 8-in. and a 4-in. seam. This ore was said to yield from 30 to 40 per cent., when the furnace-men knew how to handle it. It was sold at the mine for 30 cash per 100 catties (1s. per ton) and cost about the same to haul it to the furnace, 20 li away. The miners distinguish a number of different strata of ore here, as at Yin Ch'eng, and name them in a similar manner, but the names are somewhat different. They call the best ore "Wu Hua Kung."

The miners and officials all say that much of the best ore has been worked out, and that in former times the seams were

FIG. 1.



IRON-FURNACE AT YIN CH'ENG.

FIG. 2.

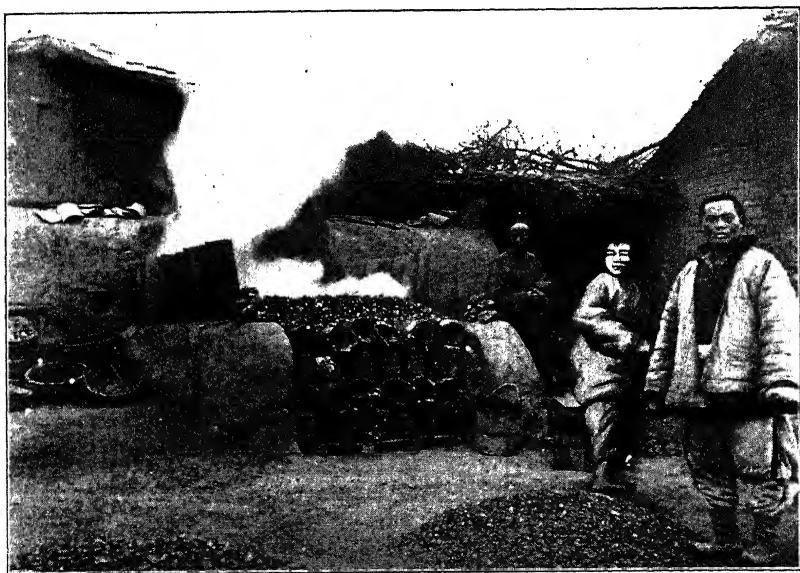
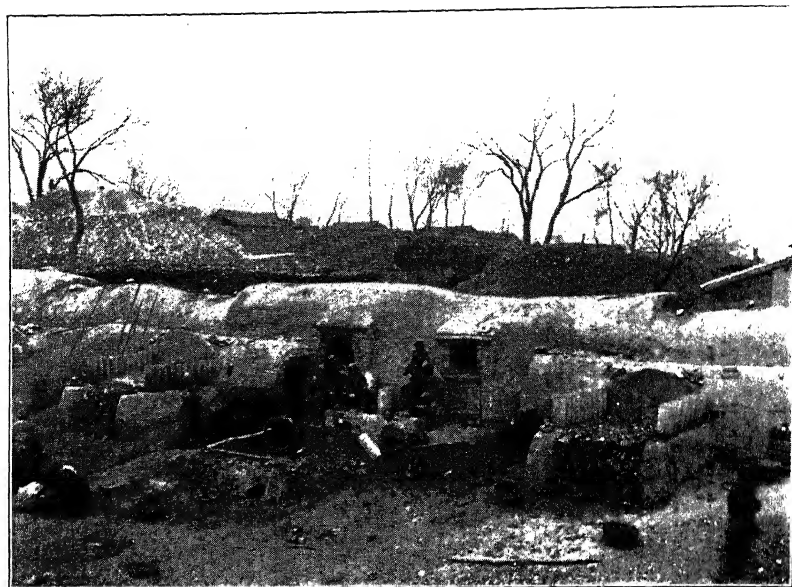


FIG. 3.



IRON-FURNACE AT TA YANG.

FIG. 4.



SLAG-DUMP OF IRON-FURNACE AT TA YANG.

thicker and of better quality. All the ores seen were impure hematites and limonites, frequently concretionary in structure, and in some cases showing plant-remains. According to the furnace-men they yield from 20 to 50 per cent. of metallic iron. The beds are about a span thick on the average, the ore not being continuous but occurring in bunches. In spite of the long time the mines have been worked there is still plenty of ore to be had, many good mines having been abandoned because of bad air or water.

At the Nan Shan P'o mine, 13 li southeast of Kao P'ing, the shaft was 60 ft. deep, and was within 100 yd. of the mouth of a coal-mine. There was only about 50 ft. difference in level between the two, the iron-ore lying that much below the coal, as at Yin Ch'eng. The iron-ore is found for the most part in irregular ellipsoidal nodules in the same soft clay-shale as at Yin Ch'eng. The nodules apparently lie in one plane, but are separated by a few feet of barren material. Those that I saw did not exceed 50 or 60 lb. in weight. The ore was the Wu Hua Kung, and was said to sell for about 4s. per ton delivered at the furnace.

Another mine, 13 li west of Kao P'ing, was opened by an adit several hundred feet long. Here the ore was a continuous stratum 4 or 5 in. thick, in the sandstone. It was P'ai Kung and sold for 40 cash per 100 catties (1s. 5d. per ton) at the mine. Another mine seen had similar ore, and the miners paid a royalty to the owner of the soil of 10 cash per cart-load of 300 catties (1.5d. per ton). There are many mines to the north and east of Kao P'ing, but I could not hear of any that were particularly different from those I saw, though some of the ore that came from about 50 li east of town was said to be specially good.

I visited several of the iron-furnaces near Kao P'ing. There were 8 or 10 of the furnaces here of the same kind as those at Yin Ch'eng, and the process is exactly similar. At one furnace, at Tung Nan Chuang, the ore cost from 70 to 120 cash per 100 catties (2s. 5d. to 4s. 2d. per ton); the coal-dust used for mixing with the ore, 30 to 40 cash per 100 catties (1s. to 1s. 4d. per ton); and the large coal, 40 cash per 100 catties. The crucibles were of the same size as at Yin Ch'eng, and were cooled and broken when cold. (See Fig. 2.)



The cast-iron was being made into wrought-iron at the time of my visit. This was done in a furnace of extreme simplicity, merely a cubical opening of 2 ft. side-measure made in the floor. This furnace is heated by a wood-fire urged by a windbox blown by two men; when the furnace is hot enough, it is partly filled with wood, the iron cakes are placed on this, the furnace is filled up with wood, and the opening of the furnace, which is about 4 by 7 in., is covered with cakes of cast-iron. When the iron is at white heat, it is taken from the furnace with long tongs and hammered lightly and quickly on an anvil by two half-naked men with hammers that do not weigh over 3 lb. By this means a small quantity of slag is pounded out; the pieces of cast-iron are welded together into billets about 10 in. long by 3 in. in diameter; these billets are made into bar-iron, usually of rectangular section, and in this form the best iron sells for 70s. per ton.

The miners at Kao P'ing get very small wages even for China. We saw men working who were only getting from 30 to 40 cash per day (about one penny), but this was only during the winter when there was no work on the farms. The furnace-men are somewhat better paid, the head furnace-man where the wrought-iron is made getting 18*d.*; ordinary workers, 3*d.*; and hammer-men, 6*d.* per day. This furnace at Tung Nan Chuang turned out about 130 tons of wrought-iron per year. The proprietor told us that his family had been carrying on the iron business there ever since the T'ang dynasty, which ended 923-934 A.D. His estimate was that about 4,000 tons of wrought-iron are made per year in the Kao P'ing district. This was an exceptionally intelligent and obliging man. Some of the other furnace-men were too frightened at the foreigners and the officials we had with us to give any accurate information; or else they thought the quickest way of getting rid of us was to tell us as many lies as possible in the shortest time.

I visited only one coal-mine at Kao P'ing. Coal is so abundant that it did not seem necessary to spend much time over the coal-mines. In the mine I saw, the seam (18 ft. thick) was evidently the same, and the coal in all respects the same, as that of Yin Ch'eng. This mine was opened by an adit 500 ft. long; the coal sold for 50 cash per cart-load of from 300 to 400 catties, or about 6 to 8*d.*, per ton at the mine.

*Ta Yang*.—This place is only 55 li from Kao P'ing; but owing to bad and narrow roads we were only able to go 40 li (13 miles), and it took the carts all the next day to go the remaining 5 miles. A very good instance of the difficulty in getting information in China was offered by a group of natives we passed about 35 li south of Kao P'ing. Seeing some slag on the road, we asked them if there was not an iron-furnace near. They all insisted there was no furnace nearer than Kao P'ing, and never had been; yet not more than 50 ft. from where they stood were several furnaces in active operation, but hidden from us by some buildings. After we had found these, we asked the men why they had lied to us; they said, it was simply to get us to move on and not to stop, for they feared if we stopped something might happen and they would be responsible for any trouble. As nearly as I could find out, the people were fully as much afraid of the officials and yamen runners we had with us, as they were of the foreigners in the party.

Ta Yang was once a very prosperous town, and needles were made here for nearly all China, but not one has been made for a long time. The place is specially noticeable for the great number of two-story houses; at present the upper stories are nearly all vacant. The appearance of portions of the place is very European and reminds one very much of some of the Italian towns.

The iron-works here are much more extensive and business-like than at Yin Ch'eng or Kao P'ing. In the large yards of the furnaces, the great piles of iron-ore, coal, yellow clay for making crucibles, soft "black dirt" (decomposed coal), which is used for mixing with the ore, and the large stacks of wood for making the wrought-iron, give a good idea of the importance of the industry; and its age is rendered evident by the immense piles of slag and cinder, which it must have taken hundreds of years to accumulate. (See Figs. 3 and 4.)

The bad state of the roads made it difficult to see the iron-mines, which, moreover, owing to the stormy weather, were not working. The first place visited was Lien Chuang, 10 li south of Ta Yang; here were dozens of shafts, said to be 60 ft. deep on the average. The one I went down passed first through 10 ft. of hard limestone with flint band, coal 2 in., soft sandstones

and clay-shale 50 ft. The ore here seemed to be a flat vein, with walls not well marked. The width was irregular; the widest place I saw was 3 ft., but it soon narrowed down to a few inches. The ore is impure hematites and limonites, and the miners distinguish two kinds, red and yellow. There is said to be only one stratum, or vein, of ore; here I consider it a vein, because of its position with regard to the enclosing rock, and because it contains quartz crystals. The ore sells at the mines for 100 cash per 320-catty cart-load (say, 1s. per ton), and the miners can make 10s. to 12s. per month. These miners had seen ore 6 ft. thick, and had heard of one place in the district where the ore had been found 20 ft. thick; but neither of these places were then to be seen.

I visited the mines of Kou Nan, 5 li south by east of Ta Yang. Here, as at Lien Chuang, there are many shafts, mostly abandoned; the ore is black ore, and is so called by the miners. It occurs in the same manner as at Lien Chuang. The best place showed 3 ft. of ore, but they were working ore as thin as 6 in. The shaft was 50 ft. deep, and there were several hundred feet of drifts from the bottom.

The mines of Sung Chia Shan, some 10 li east of Ta Yang, I visited on a very snowy day; the shaft was 100 ft. deep, the upper 30 ft. of soft earth, then 20 ft. of limestone, then 50 ft. of clay-shale. The ore was "red" ore and was a quite pure hematite, the best I saw at Ta Yang, but the furnaces pay about the same price as for the other ores, and the yield is said to be no greater. The ore was in irregular flat veins varying in width from 6 to 28 in.

These mines, according to the testimony of the oldest inhabitant of the village, a man 62 years of age, were started something over 300 years ago, and have been worked continuously ever since. There is only one stratum of ore at Sung Chia Shan. The best ore he had ever seen was during the reign of Tao Kuang (1821-1851), and was 20 ft. thick and from 80 to 100 ft. long. As a general rule, the ore-bodies here do not extend over 200 ft. in length; now-a-days the thickest ore found is about 3 ft., and the easily-accessible deposits have been worked out. A mine could be opened for about 400 tiao (£60); the mines are worked only during the time when there is no work in the fields, and the pay of miners is very

small, certainly not over 100 cash (3*d.*) per day, and probably less than that.

West of Ta Yang, beginning in the town itself, are great piles of slag and cinders fully a mile long and 100 ft. high, the result of the working of a number of iron-furnaces for many years, the furnaces still being in operation. The Chinese say that the furnaces here were started 80 or 100 years ago. There is a small furnace called Yung T'ai Shan, near the town, employing a few men, and turning out a small amount of iron annually. Farther west are the large works of Chi Hsing Shan with 15 furnaces, all running, and yielding 1,400 tons of iron per year. The furnaces are similar to those at Yin Ch'eng and Kao P'ing; the floor of the furnace is covered with anthracite; the crucibles are placed on this; the spaces are filled in with smaller coal; coal is placed on top, and over all is put a covering of sifted cinders and old crucibles. Each furnace makes one heat a day; it is charged at 10 a.m., blast is kept up for 6 hours, and the crucibles are taken out at the "first cock-crow." The ore is regarded as all of one kind, but three varieties are recognized—"red," "black" and "yellow." The cast-iron cakes from the smelting, weighing about 10 catties (13.33 lb.), are heated red-hot, plunged into water, and then broken in pieces by hammers, before being placed in the furnace where they are made into wrought-iron. This furnace and its operations are the same as at Kao P'ing. The small pieces are welded with light hammers to billets weighing about 10 catties, which are made into flat iron of small size (say, up to a section of 1 by 3 in.), by four men, three striking with 15-lb. hammers, and one holding the iron. The operation involves about four re-heatings. When the bar is finished it is stamped with the mark of the works.

NOTE.—Von Richthofen (*op. cit.*) says, that the iron is poured out of the crucibles while hot, and that much of the quality of the finished iron depends on the judicious mixtures of various kinds of iron, or whether it was poured or allowed to cool in the crucibles. I made careful inquiries about this and could find no evidence anywhere in Shansi that the iron was ever poured out from the crucibles. The crucibles are invariably allowed to cool and then broken and the iron is found either in a cake at the bottom of the crucible or else disseminated throughout the slag, as at Yang Cheng and Yu Hsien, where they are not able to melt to a cake. At Ta Yang a good deal of iron is found in flat sheets that look as if it had been poured while hot, but this comes from crucibles that have been broken in the furnace and from which the iron has run out. Von Richthofen says, also, that

two layers of crucibles are put in one furnace; this certainly was not done in any of the places I visited.

Iron is made at a number of places in the vicinity of Ta Yang; but I could learn of no other of equal importance. Various estimates of the annual product were given to me, the lowest being 4,500 tons, and the most reliable that of the district magistrate of Feng T'ai Hsien (Tse Chou), made for the whole district, including other places than Ta Yang, which was about 13,333 tons. According to the furnace-men, the ore cost 2s. per ton, and yields from 20 to 40 (average 33) per cent. Wages average 6*d.* per day; coal costs 2s. 10*d.* per ton; wood is 5s. 10*d.* per ton. The cast-iron was sold for 19s. 4*d.* per ton; wrought-iron £3 1s. per ton.

There are numerous coal-mines about Ta Yang. Three li south of the town are a number of shafts working in the soft decomposed coal used to mix with the iron-ore as a flux. They are from 50 to 120 ft. deep. The coal is about 10 ft. thick. The same seam is found NW. of the town, where at a greater depth it is 30 ft. thick. This soft coal sells for about 3.5 or 4.5*d.* per ton at the mines. The Chuan T'ai Shan mine, a few li NE. of the town, lately opened by a man from Tse Chou, is a very creditable piece of work for any country. There are two shafts, one for hoisting, and the other for ventilation and for water. The small amount of water is hoisted in cow-hide buckets. This mine is on ground leased at £30 per annum. I went down the main shaft, 240 ft. deep, riding astride a short stick, in the usual manner. The shaft is vertical, about 100 in. in diameter, and bricked for 100 ft., till it reaches the solid rock. The coal is hoisted by a large windlass worked by eight men. At present they are turning out 400 baskets of 250 catties each per day (say 60 tons). There are 50 miners underground, paid from 200 to 240 cash per day, and the foreman is paid 320 cash (9.5*d.*) per day. A large coal-fire was burning at the bottom of the shaft; and the mine was very well ventilated. The coal was 30 ft. thick, but they were working only the lower layer of 6 ft., above which there was a natural parting. The coal left overhead stands remarkably well, supported by 8-in. stulls placed about 10 ft. apart; the galleries are about 15 ft. wide; the seam has a slight dip away from the hoisting-shaft. The best coal is sold in lumps of from 50 to 100 lb.,

being hauled to Ta Yang in small carts, carrying from 400 to 500 lb. each. The lump-coal sells at the mine for from 2s. to 2s. 7d. per ton; the small coal being largely carried away by young boys in baskets. The coal is very clean, shows hardly any pyrites, and burns with a long blue flame. The tools used by the miners consist of iron bars 6 ft. long by 1.5 in. in diameter, small picks 6 in. long, and iron wedges. It was a pleasure to visit this mine after crawling on hands and knees through the narrow and crooked drifts of the mines at Yin Ch'eng and Kao P'ing. Here the drifts were 6 ft. high, about 15 ft. broad, and as straight and workmanlike as possible. The miners seem contented with their lot. They say they begin work at dawn and stop at midnight; but they probably rest during part of the day; for I saw a number asleep near the big coal-fire and one was smoking opium. The coal is brought to the shaft in baskets on sledges drawn by men. There is only one workable seam of coal here, but there are several other seams of a few inches' thickness.

*Tse Chou.*—On the road to this place, about 10 li south of Ta Yang, there is a small iron-works, making about 300 tons per annum. Just before reaching town we met large numbers of camels, employed in carrying salt and soda from the west, and grain and beans from the north; they are also largely used for local traffic. Large numbers of camels are used in Shansi; but as they usually travel at night we did not often meet them. At Tse Chou, there are no iron-mines or furnaces, but some foundries. I visited one, in which were made pans for boiling water. These pans are shipped as far as Manchuria; and probably nearly all northern China uses pans made in Shansi. I bought one 23 in. in diameter, weighing 16 catties, for 200 cash. The market price is about 42s. per ton. The molds are made of fine sand over a backing of coarse, and are dried in an oven cut in the loess. The subsequent casting is done out of doors.

While at Tse Chou I inquired about iron-mines to the south and southeast. Some mines were reported, but they were either shut down or else were new mines not yet opened; and no furnaces were known in that direction.

I visited the largest mine in the vicinity of Tse Chou, near the road to Ta Yang, and called the Ch'uan T'ai Shan. This

mine, which had been in operation for about 13 years, has two vertical shafts 240 ft. deep, 8 ft. in diameter, and 100 ft. apart. The mine is operated by an association of miners who seem to have no head-man or manager. They hire the ground (about 6 acres) at an annual rent of from £60 to £70. The coal-seam, 22 ft. thick, is made up of 4 ft. of "mei," a soft coal which is the bottom layer; then 12 ft. of "t'an," or hard solid coal; and above this 6 ft. of "ma shan," which is also a solid coal, but which has a parting seam separating it from the "mei," and in which the jagged fracture that seems characteristic of Shansi coals is specially developed. The coal of Tse Chou is noted all over southeastern Shansi for its excellence. It is a pure anthracite; some of it has a dull appearance and fracture, while much of it is a brilliant shining black, and is so compact that it will not soil a clean white handkerchief when rubbed. It burns with a slight bluish flame and no smell, and seems a free-burning coal, for an anthracite. In selling this coal the miners make three classes:—T'an, or large lump-coal, sold at 3s. 3d. per ton; Sui T'an, or mixed coal, sold at 1s. 9d.; and Hsi Mei, fine coal, sold at 1s. 4d.

The method of mining is ingenious, and, though wasteful of coal in that it does not extract all the coal from a given block of ground, it is very sensible under the circumstances. A series of rooms, separated by from 10 to 20 ft. of solid ground, is opened. The size of the rooms varies from 15 by 20 ft. up to 35 by 60 ft., depending on the character of the coal, and of the sandstone roof, being larger with the harder coal. There are over a hundred of these rooms in the mine. The dimensions of the room having been decided on, a cut is made completely around it and is carried up to the "ma shan," 16 ft. from the floor; at the same time the soft "mei" is taken out under this by a series of cuts 46 in. by 38 in. high. When the coal begins to crack—to "talk," as the miners say—they stop work in these drifts, and the block soon comes down, parting at the "ma shan" seam, which is taken down later.

We saw one block of coal, just fallen, which was 50 by 30 by 12 ft. in size, or about 750 tons. Wages are paid per block of coal excavated 48 in. wide by 38 in. high by 30 in. long. In the lower seam, the miners get 200 cash for each of these, and cut about two per day; including lost time they are said to make

seven taels, or about £1, per month. After the "ma shan" seam has been taken down, the roof (which, as everywhere in Shansi, is sandstone) caves in rather large blocks, some of the rooms being fully 50 ft. high. The large openings in this mine, plainly showing the 22-foot seam of coal in contrast with the lighter sandstone above, were most impressive. A particularly weird sight was a big room, 60 ft. in diameter by 30 ft. high, near the shaft, where a monstrous fire was glowing; on a line stretched between two poles some rags of clothes were drying, and men were lying all around, some asleep and some smoking opium. Another curious feature was the large number of rats, fed with millet by the miners, and believed to give warning of accidents.

Leaving Tse Chou, we passed a number of coal-shafts, and shortly after came into a limestone region. On the top of the pass we found a number of small shafts and openings from which iron-ore had been taken, but the villagers near would give us no information about them. There were some old slag heaps near Chou T'sun Chen, and their great extent shows that iron has been made largely here at one time, although at the time of my visit no furnaces seemed to be working.

About 10 li before reaching Yang Ch'eng, we passed an extraordinary collection of workshops in a little nest in the mountains, where they were making shovels; a portion of our party went by another road and saw a similar village, where nothing but scissors were made, the workmanship of both the scissors and the shovels being very coarse.

*Yang Ch'eng.*—This small city, with stone-paved streets, is a center for the iron-works of which there are a number in the neighborhood; but it is a much less important place than Ta Yang. I visited a works 10 li NW. of the town. There is some slight difference in the details of the making of the iron here, but the general process is the same as at Yin Ch'eng, etc. The most noticeable fact is that they are not able to make a good fusion; they get cakes of iron sometimes, but much of the iron is mixed through the slag from which it is separated by pounding with hammers. The furnaces are rather larger, the floor measuring 128 by 52 in., with the two side-walls 42 in. high; the crucibles are 8.5 in. in diameter and 22 in. high. The furnace mixture is made with 1,300 catties of iron-ore



(costing 3s. 6d. to 7s. per ton); 500 of small coal, and 500 of coal-dust ("hei tu"). It takes two days to melt a charge, and the yield is from 500 to 700 catties of iron. Wrought-iron is also made here; but the iron-works are much less extensive than at Ta Yang, and the people are less prosperous. We had much trouble in finding the iron-mines at Yang Ch'eng; it was only after we had sent away the yamen runner, who, I think, frightened the people and tried to get money from them, that we were able to find a mine, opened by an adit with several branches, aggregating several hundred feet in length, and all in soft sandstone. Here they were working on small bunches of ore about 3 ft. long by 12 in. thick, which were said to be a good size, and not often larger in this mine. At some of the other mines bunches of from 6 to 7 tons had been found. The ore sells for from 4s. 6d. to 5s. 3d. per ton. There must be considerable iron made about here; the district magistrate estimated 2,000 tons annually; but I think there must be more, since a large number of places were reported as having furnaces. For the benefit of any future visitor, I give the following list of places, furnished to me by the district magistrate. The distances given are all measured from Yang Ch'eng.

*List of Silver-Mines South of Yang Ch'eng Which Have Been Reported or Petitioned About.*—1. P'an T'ing Ho in the Nai Nai mountains, 90 li; 2. Shang Shan Lin, Tung Kou, Tung P'o, 50 li (1 shaft here); 3. Yang Ling Miao at Yin Tung Shan, 40 li; 4. Sheng Wang P'ing, a well-known place; Kuei Lao Tung, a small place near this, 90 li; 5. Yuan Shan Kou, half way to Yang P'o Shan, 120 li.

*List of Furnaces and Coal-Mines not Making Iron but Making Castings from Iron Purchased Elsewhere.*—1. Yin Chia Kou, 10 li; 2. Mei Ch'uan T'sun, 10 li; 3. Hao Yu T'sun, 20 li; 4. Shang Kung Chai, 10 li; 5. Shan Tou Hui Ch'in, 7 li.

*List of Coal-Mines West of the City (There are also Furnaces Here).*—1. Li Ch'iu, 8 li; 2. Yu Kou, 15 li; 3. Ke Liao Kou, 40 li; 4. Chang Chuang has only coal-mines.

*List of Foundries.*—Liu Shan T'sun, 20 li east of the city; and Nai Yang, 10 li; Ling Hou, 15 li; and Hsieh Kou, south of the city.

Inquiring about the silver-mines, I learned that they were

worked during the Ming Dynasty, in the reign of Wan Li, 280 years ago, and are said to have been very rich; the miners making so much money that they turned the mountain into a sort of Venusberg, taking their women into the mines and holding high carnival. There was much fighting, with disorders of all kinds, which gave the authorities so much trouble that they ordered the mines to be shut up; and though permission to open them has often been demanded, it has never been granted. I sent a man to examine and he reported that no shafts or old openings are known. He brought back as silver-ore some specular iron-ore and some yellow dolomite. There are various petitions on file in the Yamen at Ying Ch'eng in reference to these mines.

Many coal-mines have been opened by adits about Yang Ch'eng. We saw, 10 li NW. of the town, several adits which have been working for the last 70 or 80 years and are said to extend from 7 to 8 li in the mountain, on a seam of coal from 3 to 6 ft. thick. The large coal sold at the mouth of the adit for 10*d.* per ton and the small coal and "black dirt" for 6*d.* per ton. There are several seams of coal reported by the natives to be from 5 or 6 up to 20 ft. thick. It is an anthracite of good quality. At some of the other mines it sells for 6*d.* and 10*d.* per ton. At town, the price is 2*s.* per ton for large coal and 1*s.* 2*d.* for soft. The seams can be readily opened by adits, as they are all above the level of the valley.

Among the other articles made near Yang Ch'eng are iron scissors, manufactured in small shops where several men work together. One man can make 10 pair per day; and they sell for about 1½*d.* each. The files used in making these scissors are of native manufacture, and come from Lou Ho Kou, in Hounan.

After leaving Yang Ch'eng, we passed a number of coal-mines. At the point, about 13 li from town, where we ascended from the valley into the upper sandstones, there were a number of coal-mines on the opposite side, working on at least three distinct seams of coal. We did not see any more coal-mines, or see or hear of any iron-mines or furnaces near our road, until we arrived at Ling Shih.

*Ch'in Shui Hsien.*—This is a miserable little city, reached by traversing a barren country. The people are wretchedly poor, having never recovered from the famine of 1878.

There are no coal-mines at Ch'in Shui; but about 70 li to the southwest are both coal-mines and iron-furnaces. The district magistrate gave the following statement:

*Furnaces.*—Twenty-third year of Kuan Hsu (1897): Number of shops established, 131; iron produced in a year, 55 wan (380 tons). Poor shops shut after a short existence, number 8, produced 5 wan (35 tons).

*Coal-Mines.*—Twenty-third year of Kuan Hsu: First-class mines, number 7, produced hard coal during the year, 120 wan. Soft-coal mines, number 17, produced just enough coal for local consumption. Second-class mines, number 14, produced 70 wan during the year.

These figures are those sent to the Governor at T'ai Yuan and the actual production is probably many times that given; I did not visit mines or furnaces. The coal was anthracite, apparently of very good quality.

Leaving Ch'in Shui we crossed the Wu Ling passes, which are well described by von Richthofen. The road or rather mountain-trail was something awful, being a river of water in some places and a porridge of sticky loess-mud in others. The next day, we were in a very beautiful and fertile valley.

*Yi' Ch'eng.*—This place, where we took tiffin, is a prosperous town for this part of the world. We saw several large shops with good-sized stocks of iron from Yin Ch'eng, Feng T'ai Hsien (Tse Chou) and Yang Ch'eng. There are coal-mines not far from Lung Hua, where we stopped over night. The coal used there came from 30 li to the northeast and sold for 200 cash per cart-load (2s. per ton). The coal used at Yin Ch'eng comes from mines 50 li to the southeast and sells for 4s. per ton in town. This coal was all anthracite of splendid quality; nobody seemed to know much about the mines, except that they were opened by adits and not by shafts.

*P'ing Yang Fu.*—At Shih T'sen, 60 li south of this place, we first saw the bituminous coal and coke. The coke sells for from 28 to 32s.; the coal for 6s. 10d. per ton. They both come from the mountains to the west, the mines being 60 li distant.

At P'ing Yang Fu, we found the coal and coke as at Shih T'sun. The best mines were said to be about 100 li west. There are no iron-mines or iron-furnaces near here. The iron all comes from the Tse Chou region and sells at from 133 to

175s. per ton for wrought-iron. We did not visit any of the coal-mines. The missionaries report the seams as very large, from 30 to 40 ft. being the width stated, with a maximum of 60 ft. After leaving P'ing Yang, we saw coal and coke in the towns and villages everywhere, all bituminous and coming from the west, selling for from 8 to 9s. per ton, and costing 1s. 7d. at the mines, which were all said to be worked by tunnels. Everyone we questioned—missionaries, merchants, coal-packers and officials—told the same story of the great quantity of coal in these mountains to the west of the valley of the Fen river where we were traveling. We crossed the well-known Ling Shih or Han Hsin Ling pass, named after the general who cut the road through, about 200 B.C. The altitude here is about 4,500 ft.; beyond the pass we saw a number of small coal-seams and there are many mines near here; the coal is bituminous, is very black and glistening; it sells for 50 cash per donkey-load of 150 catties (16d.); coke is also burned here, sells for 8s. per ton.

*T'ai Yuan.*—From this place we visited the Wang Fang Shan mines, 30 li away. In the mountains, 20 li from town, we came to a sulphur-mine, where a hundred men or so were making sulphur from iron pyrite, which occurs in beds in the clay-shale. This sulphur is used in the arsenal at T'ai Yuan for powder-making. At 30 li from T'ai Yuan we saw an iron-furnace. The process here was quite the same as at Yin Ch'eng, etc.; but the furnace was made by cutting a trench in the loess, 27 ft. long by 6 ft. wide, and the windbox was at the end of the trench. The pots for melting the iron were much taller than those previously seen, being 44 in. high and 6 in. in diameter; 200 pots were placed in the furnace at once. They were not able to get the iron in a cake here, but it was mixed through the slag, as at Yang Ch'eng. The coal was bituminous, and cost about 1s. 4d. per ton. Wrought-iron was not made at this furnace; but was made at other works further west, which I had no time to visit. The iron-ore came from beds at the base of a cliff, half a mile south of the village; the ore being exposed for a length of fully 1,000 ft., and worked in from 20 to 30 places. It was all impure hematite and limonite, but more continuous than in the places previously seen. The bottom-bed was 100 ft. in length and averaged 4 in. in thick-

ness. Above this were bunches of ore 10 in. thick by 24 ft. long. The ore sells for from 2s. 9d. to 7s. per ton. The mines were leased, the miners paying the owner about 40 per cent. of the value of the ore; his leases were bringing him in about 300 cash per day at the time of my visit.

In a bed near these iron-mines I found a large number of fossil shells which had been partly converted into an impure iron-ore. These were identified by R. Bullen Newton of the British Museum as *Productus semireticulatus*, and *Orthothetes crinitula*, var. *senilis*. Other fossils from Shansi were identified as a *Productus amplexus* (a coral), *Fusulina* (a foraminifer), *Productus longispinis*, *Orthis* (like) *Michelini*. All these fossils are supposed by von Richthofen to be sub-Carboniferous.

*Yu Hsien.*—From T'ai Yuan to Yu Hsien, the road was nearly always in loess; and we saw no coal-mines until, a few hours before reaching Yu Hsien, we passed a couple of mines producing anthracite, which sold at the mines for 1s. 2d. per ton for small, and 4s. for large lump-coal. There were several seams of coal here, the largest 4 ft. thick. In the valley in which Yu Hsien is situated, there were large numbers of lime-kilns, burning the pebbles from the stream and the lime rock. At Yu Hsien the coal occurs in five or six seams, the only one worked being 18 ft. thick. The other seams are good; but are left unworked, as they get all the coal they want from the 18-ft. seam. The coal sells at the furnaces for 4s. 10d. per ton.

The furnaces for making iron at Yu Hsien are 15 ft. long by 6 ft. wide, with walls 64 in. high (Figs. 6 and 7). The pots here are more refractory than at any of the other places visited. They are made 42 in. high by 5.5 in. in diameter (Fig. 5), of a mixture of 17 parts of clay and 3 parts of pounded slag. Each furnace holds more than 200 pots, containing 3,000 catties of ore and 800 of soft coal. Each crucible gives about 10 catties of iron, which does not melt, but remains in the slag, from which it is freed by hammering. It takes 3,000 catties of coal to melt the iron. Wages here average 6d. per day; ore costs 4s. 10d.; soft coal 3s. 2d. per ton; clay 2s. per ton, used for making pots for smelting the ore. Wrought-iron, which is made in smaller furnaces (Fig. 8) than those at Ta Yang, sells at from 70 to 105s., and cast-iron from 24s. 6d. to 28s. per ton. The coal here seems as if partly bituminous

FIG. 5.



·MAKING CRUCIBLES AT YU HSIEN.

FIG. 6.



FIG. 7.



IRON-FURNACE AT YU HSIEN.

FIG. 8.



WROUGHT-IRON FURNACE AT YU HSIEN.

in character; and what is called "shih t'an" has much of the appearance of cannel coal. There is a red iron-ore used here in small quantities, which comes from 50 li north at Pei Hsien, and is said to sell for 8s. 9*d.* per ton. A soft yellowish-red ore costs 3s. 6*d.* per ton.

An iron-mine about 7 li east of town, which we visited, was opened by an adit. The surface was loess; under this came a bed of yellowish-gray sandstone; next came a seam of coal a few inches thick (not worked here); and, below this, clay-shale, in which the iron-ore is found in bunches. The largest seen was 3 ft. long by 10 in. thick, and probably 2 ft. wide. Five men were working in this mine, each earning from 1 to 3*d.* per day. In three months they had taken out 8,000 catties of ore; they worked here only about five months in the year. We also visited a works where they were making the blades of hoes, the handles being put on at P'ing Ting Chou. The iron made there is said to be better than that of Yu Hsien. Leaving Yu Hsien, we traveled in the coal-measures for about 30 li. Several seams could be seen. The following section was noted at 20 li from Yu Hsien:—

	Feet.
Loess, . . . . .	25
Coal, several seams, . . . . .	20
Shales, . . . . .	5
Sandstone, . . . . .	20
Shale, . . . . .	40
Coal, . . . . .	6
Soft Clay-Shale, . . . . .	10
Sandstone, . . . . .	15

Shortly after this we passed numerous abandoned iron-mines; the ore was said to have been nodular ore, much the same as we saw at Yu Hsien. At a small village, called Kuan T'ou, where we stopped for tiffin, we saw some semi-bituminous coal. The mine is at Ch'u Ch'ü Ch'eng, said to be 35 li due south, and worked through shafts 130 ft. deep; the seam being from 12 to 18 ft. thick. Coal sells at Kuan T'ou at 4s. per ton for large lumps and 1s. 9*d.* for small coal. They are said to make coke which sells for from 12s. to 16s. per ton. After leaving here, we went up 500 ft. to an altitude of 5,000 ft., and then descended the difficult Shih Pa P'an (18-Bend Pass), by a very steep and crooked trail, for a distance of perhaps



4 miles, to a small river of clear water at an elevation of 2,500 ft., at Sha K'ou, where we stopped for the night. The coal here comes from Yu Hsien and sells for 12s. per ton: at Mao Er Shih, farther on, and 120 li from Yu Hsien, the same kind of coal was selling for 28s. per ton.

#### PRODUCTION OF COAL AND IRON IN SHANSI.

There are no trustworthy statistics to be had of the coal- and iron-production of Shansi. The coal-output is certainly very large; coal is used everywhere, and the great number of men and animals employed in mining and transporting it make the industry the most important, next to agriculture, in Shansi. The consumption per family is stated by the Chinese to be fully 10 tons per annum, which, if we take the population of Shansi at 15,000,000, or 2,000,000 families, would give a consumption of 20,000,000 tons per annum, which I think is beyond doubt much too large. It is equally certain that an estimate of 1 ton per annum per family, or 2,000,000 tons, would be much too small; and the true amount lies somewhere between these wide limits. My own opinion is that 5,000,000 tons per annum is fully as likely to be under as over the mark.

With regard to iron-production there is the same difficulty of absence of statistics. I make the following estimate more to give an idea of the production than with any pretence at accuracy:

	Tons Per Year.
Yu Hsien, 60 furnaces working 250 days per year, making 500 catties per diem each, . . . . .	4,500
P'ing Ting Chou, 250 furnaces, each making 500 catties per diem, . . . . .	18,000
Yin Ch'eng, estimate based on conversations and information obtained while there, . . . . .	6,000
Kao P'ing Hsien, statement of furnace-men, . . . . .	4,000
Tse Chou Fu, includes Ta Yang, etc., statement of magistrate based on file in his yamen, . . . . .	13,333
Yank Ch'eng, statement of magistrate, . . . . .	2,000
Ch'in Shui, statement of magistrate, . . . . .	415
T'ai Yuan Fu, information obtained from furnace-men, . . . . .	2,000
Total, . . . . .	<u>50,248</u>

I think that this is a minimum estimate, and that it is perfectly safe to say that more than 50,000 tons of iron are now being made in Shansi per annum.

When von Richthofen was in Shansi, he estimated the production of iron at 160,000 tons per annum, which was considered an absurdly large estimate by critics who had never been in the province, but I have no doubt he was well within the truth. The district magistrate at Tse Chou said that the iron made in that district now was only one-fourth of what it was 30 years ago, which was about the time that von Richthofen visited the province (1870-72). If the iron-trade has declined as much in the rest of the province as it has here, my estimate and von Richthofen's would not be so very different.

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### The Box Electric Rock-Drill.\*

BY FRANK E. SHEPARD, DENVER, COLO.

(Albany Meeting, February, 1903.)

#### INTRODUCTION.

ELECTRIC power in mining-operations is now successfully applied to haulage, hoisting, lighting and pumping; and until lately, drilling was the one department of mining in which an electric source of energy had not been introduced. Drilling by electric power has been the field for much investigation and several forms of electric-drills have been tried; but in the severe service of rock-drilling many difficulties have been encountered, and the heavy cost of repairs has caused a deep prejudice against this system of drilling.

In the majority of electric rock-drills previously placed on the market, two principal factors have been considered; first, a portable electric motor usually of heavy pattern, coupled to the driving-mechanism of the drill proper by means of a flexible shaft; and second, the coupling of some form of reciprocating cross-head to a piston by means of a spiral spring, the drill-bit being fastened to the piston-rod by some suitable form of chuck with fastening bolts and nuts. In this latter form of drill, the cross-head has a fixed stroke, while the piston with drill-bit attached is allowed a variable stroke through the

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\* Invented by W. A. Box and manufactured by the Denver Engineering Works, Denver, Colorado.

medium of the spiral spring. The principal difficulties in this type of drill are as follows:—expensive repairs in the flexible shaft; breakage in the spiral springs; large space required in setting-up; and loss of time in adjustment. In the effort to overcome these objections, the Box electric rock-drill is designed with the electric motor mounted on the guide-shell, directly behind the drill proper; the troublesome spiral springs are replaced by an air-cushion; and the drill-bit, instead of being reciprocated back and forth in the hole of the rock, is simply turned in the hole, while a hammer strikes a blow on the end of the drill-bit,—similar to the manner in which a miner would turn the drill with his left hand and strike a blow with the hammer in his right hand.

#### DETAILS OF CONSTRUCTION AND OPERATION.

*Development.*—The elements of the earlier models are given in Fig. 1, which shows the cross-sections of the Model 4 drill. The electric motor, designed to be waterproof, is shown at the left, mounted on the guides of the shell and coupled to the drill proper by means of machine-cut forged-steel gears. A taper-pin, with nut, fastened the motor to the drill and the removal of this pin allowed the motor to be removed. The crank was connected to the cross-head by means of a connecting-rod; the drill-hammer was coupled to the cross-head by means of a spiral spring; one end of the hammer formed a piston which was fitted to a small cylinder solid with the cross-head; the drill-bit was held with small end-play, in a revolving chuck driven by a train of gearing from the crank-shaft.

“Model 6” drill is the latest improved form, the important change being the replacement of the spiral spring by an air-cushion. The relation of the electric motor, transmission-gears, crank, and connecting-rod remains practically the same as in the Model 4 drill, but Fig. 2 shows the details by which a perfect air-cushion is placed between the cross-head having a constant stroke, and the variable-stroke hammer. It will be noted in Fig. 2 that the element corresponding to the cross-head of an engine becomes a cylinder in which is fitted the hammer of the drill. This cylinder of cast-iron is machine-finished inside and out, and the forged-steel hammer is machine-finished all over and fitted to the bore of the cylinder. The form of the ham-

mer resembles the piston and rod of a steam-engine, leaving an air-space between the piston of the hammer and the heads of the cylindrical cross-head. When the drill is in motion, the air on each side of the hammer-piston is alternately compressed and rarefied, giving exactly the effect of a spring between the cylindrical cross-head and hammer, thus furnishing an element which is indestructible.

To compensate for possible leakage and to insure a full supply of air, two ports are cut in the side of the moving cylinder which are so located that as soon as the piston moves from its central position, one port is opened to the atmosphere and a supply of air rushes in; then the other port opens and furnishes an air-supply to the other side of the piston. Thus, these two ports serve to equalize the air-pressure on both sides of the piston as it passes its central position. Oil-grooves are turned in both piston and rod to furnish an oil-packing between the piston and the cylinder-walls; also between the rod of the hammer and the cylinder-head. These parts of the drill have been described in detail, for the reason that springs were found to be objectionable and certain to break; their replacement by the air-cushion seems to have solved the question perfectly. Fig. 3 shows a drill with the cover removed in order that the internal mechanical construction can be seen.

It was thought that in time, the piston might wear and destroy the air-cushion by allowing the air to leak from one side of the piston to the other side. In order to ascertain the effect of wear, one of the pistons was turned down  $\frac{1}{32}$  in. smaller in diameter and the drill then used on hard rock. No appreciable difference in speed of drilling was noted. An amount of wear equal to  $\frac{1}{32}$  in. in diameter would require several months of the hardest service.

*Rotation.*—The Box electric rock-drill is the first drill to discard the troublesome pawls, springs and ratchets found in the rotating mechanism of other types. The chuck of the Box drill is of forged steel, mounted in removable composition-bearings. A train of gears, driven from the main crank-shaft, gives a continuous rotation to the chuck by means of a gear cut on the outside surface of the chuck.

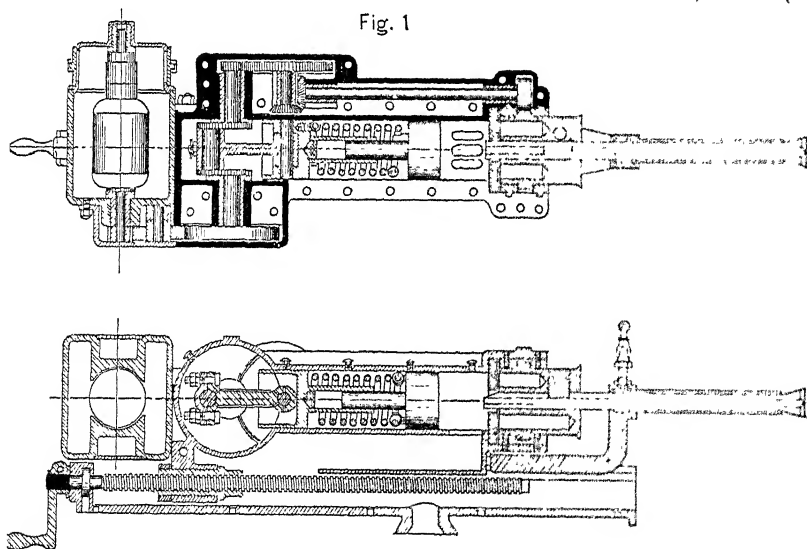
In the chuck is mounted an automatic key engaging two flattened surfaces forged in the drill-bit, which allows an end-

play in the drill-bit of about 1 in., but compels the bit to turn with the chuck. There are no bolts and nuts to loosen, and the rotating-mechanism connecting the crank-shaft with the chuck has ample strength to turn the drill-bit in any kind of rock.

*Hinged Guide-Shell.*—Another important improvement consists of a hinge between the guide-shell and the cone of the clamp, which enables the whole drill to be tipped to one side or the other, thereby enabling long drill-steels to be quickly inserted or removed from the drill, or hole, without disturbing, or changing, the set-up of the drill. (Shown in Fig. 4.) When the drill is tilted back into place, it is again in proper alignment with the hole. The guide-shell, drill and motor are mounted on a tunnel-column shaft-bar with clamp and arm (Fig. 4), or on a tripod (Fig. 5).

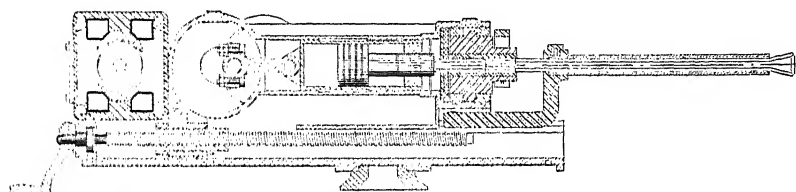
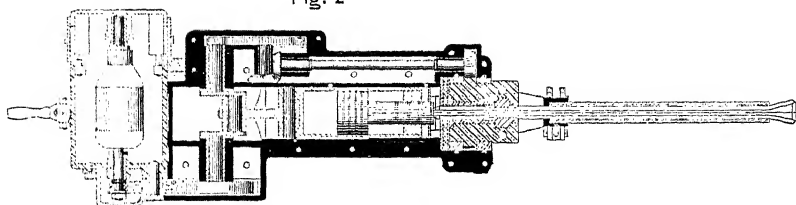
*Controller and Motor.*—The electric current is conducted from the main transmission-wires to the controller through heavy rubber-covered mains; the controller is of special design, and includes the necessary resistance for five speeds with proper fuse-blocks, all contained in a compact waterproof aluminum case. Connection is made with the drill-motor by a short heavy rubber-insulated cable. The electric motor was designed by

Fig. 1



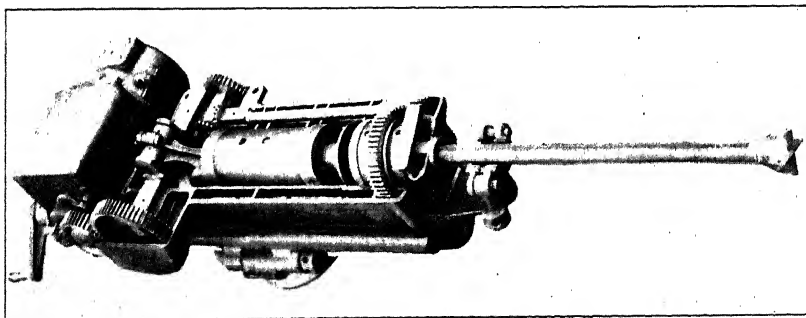
CROSS-SECTIONS OF DRILL (MODEL 4), SHOWING SPRING-CONNECTION BETWEEN CROSS-HEAD AND HAMMER.

Fig. 2



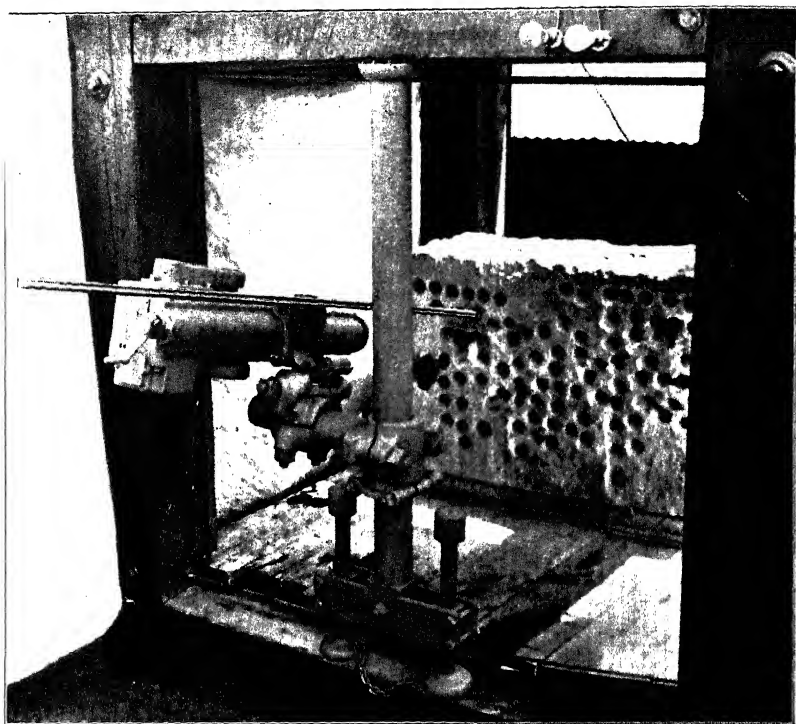
CROSS-SECTIONS OF DRILL (MODEL 6), SHOWING AIR-CUSHION BETWEEN  
DRIVING-MECHANISM AND HAMMER.

FIG. 3.



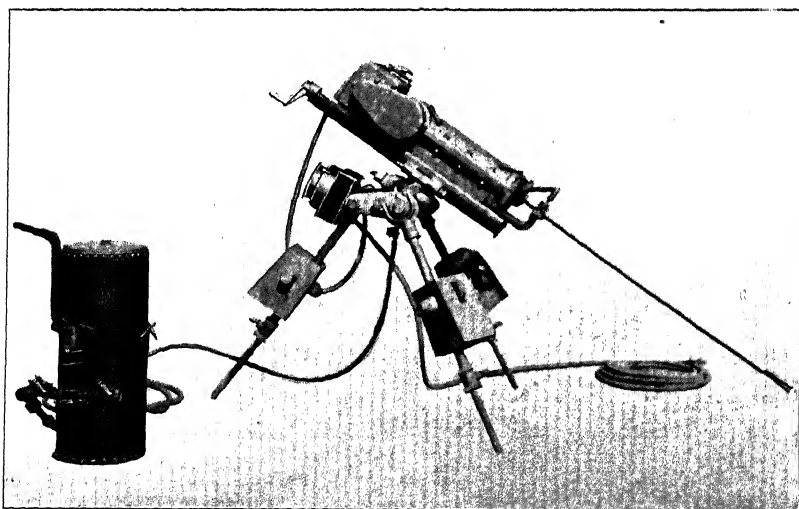
DRILL (MODEL 6), WITH COVER REMOVED, TO SHOW THE DETAILS OF THE  
OPERATING MECHANISM.

FIG. 4.



HINGED GUIDE-SHELL; SHOWING THE MANNER IN WHICH IT IS TIPPED FOR THE REMOVAL OR INSERTION OF DRILL-STEELS, WITHOUT CHANGING THE SET-UP OF THE DRILL.

FIG. 5.



DRILL (MODEL 81) MOUNTED ON TRIPOD, WITH WATER-TANK, WATER-TUBE

the General Electric Co. expressly for this drill and service, after most thorough investigation and practical tests. While every means are employed to reduce the weight and size, all precaution is taken for careful insulation and substantial construction. The motor has a waterproof case, and aluminum caps are used on pinion and commutator-ends. Suitable hand-hole plates are provided for inspection and care of the carbon-brushes. The motors are wound for either 110 or 220 volts direct current.

*Removal of Crushed Rock by Water.*—For the purpose of removing the crushed rock from the drill-hole, a long steel tube is placed over the drill-bit in order to deliver a stream of water to the cutting-edges of the bit. This tube, of an internal diameter slightly larger than the outside diameter of the drill-bit, has a groove cut on the outside surface nearly its entire length, similar to a key-way in a shaft. Another steel tube, whose internal diameter fits the external diameter of the grooved tube, is pressed over the inner tube, thus leaving a hole through the shell of the finished water-tube. One end of this tube is fastened to a bracket which is bolted to the chuck-end of the drill. In this bracket is a water-duct, which is connected by rubber hose and suitable fittings to a portable tank, holding a supply of compressed-air and water. When the water-valve is open, a stream of water is delivered with a concentrated washing-action to the point of the drill-bit, and the small rock-particles cut out by the drill are washed back from the bottom of the hole on the outside of the tube to the face of the workings.

*Dust Prevention.*—One of the most serious conditions in rock-drilling is the danger to the operators in breathing the dust produced, but its removal by water as soon as it is formed overcomes this danger. This advantage, alone, makes the additional apparatus for the water-device worthy of use, as well as providing humane methods for the comfort and protection of the drill-men. Another advantage of the water-attachment is that the stream of water, playing on the cutting edges of the drill-bit, preserves its temper and increases its life.

*Tube Serves as a Guide.*—A reciprocating type of rock-drill, meeting a seam or pocket, is almost sure to be deflected from its true course and finally stick, or "fitcher," as the miners say.



This is particularly the case in drilling through seams running in a line diagonal to the axis of the drill-hole. When the Box electric drill encounters such a condition, the drill-bit action is such that the cutting-edges rebound but a small distance from the bottom of the hole, and being supported by the water-tube to a point within about 3 in. of the cutting-edges, any sticking, or "fitchering," is prevented; even under the worst conditions but a slight deflection is produced. It is possible to drill one hole across another at a very slight angle, and have but a slight deflection in the second hole.

*Standard Size.*—The Box electric drill is made in but one size at present, which is equivalent to an air- or steam-drill with a 3-in. piston. The dimensions of the drill are as follows:—

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	Inches.
Length over all when drill is on back end of feed-screw, . . . . .	44½
Length over all when drill is at extreme front end of feed-screw, . . . . .	64½
Width of drill, over all, . . . . .	15½
Distance from center of arm to top of drill, . . . . .	17
Distance from center of arm to back end of feed-screw handle, . . . . .	29½
Distance from center of arm to front end of guide-shell, . . . . .	15
Distance from center of arm to front of drill when drill is on extreme front end of feed-screw, . . . . .	35
Length of feed, . . . . .	20

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The weights of various parts of the drill are as follows:—

---

	Pounds.
Total weight of guide-shell, electric motor and drill, . . . . .	350
Weight of guide-shell and drill, . . . . .	260
Weight of drill proper, without guide-shell, . . . . .	197
Weight of guide-shell, . . . . .	63
Weight of electric motor, . . . . .	90

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*Cost of Plant.*—A one-drill prospecting-plant, weighing approximately 4,300 lb., costs \$1,160 delivered on board cars at Denver, Colo., including the following fittings for both drill- and power-plants:

Drill-Plant.—1 Model 6, Box electric drill, with electric motor and guide-shell; 1 6-ft. tunnel-column, shaft-bar, or tripod, complete with clamp, pinch-bar and wrench; 1 portable water-tank, with attached hand-pump, hose and fittings; 1 controller, with resistance and fuse-block, encased in waterproof aluminum

case, with 50 ft. of heavy 2-wire electric cable; 1 complete set of drill-steels, including 18 pieces, varying in length from 2 ft. 6 in. to 6 ft.; 1 complete set of water-tubes for the drill-steels.

Power-Plant.—1 electric generator, direct-current, belted-type, with sliding base and rails; 1 semi-portable boiler and engine, both mounted on one cast-iron base, with fittings complete; 1 endless cemented leather belt to connect engine with generator; 1,000 ft. No. 8 copper insulated wire, for transmission of 500 ft. from generator to drill; 50 insulators for transmission line; 1 ammeter, ranging from 0 to 25 amperes; 1 voltmeter, ranging from 0 to 150 volts; 1 double-pole knife-switch; 1 fuse-block, with fuses.

A four-drill plant including machinery suitable for the increased size weighs approximately 12,500 lb., and costs \$3,750 delivered on board cars, Denver, Colo.

Where a direct-current electric-light system is employed about the mine, the drills may be run from this circuit. If an alternating-current power-system be available, the current may be transformed to 110 volts, or 220 volts direct-current system, by means of a motor-generator set. A motor-generator set, or a gasoline-engine plant, will cost about the same as the steam-boiler and engine-plant of the same capacity.

*Impact, or the Force of Percussion.*—In order to give some conception of the force of the blow delivered by a machine-drill, it is well to investigate the principles of this action. Among the mining fraternity, it is customary to measure the blow of a machine-drill in pounds, obtained by multiplying the area of the piston of the air-, or steam-drill by the pressure per square inch. This calculation is not correct, because it does not take into account the weight of piston, chuck, and drill-steel; or the distance in which the moving masses are brought to rest. If the energy stored up in a swiftly-moving hammer could be measured and this quantity, expressed in foot-pounds, be divided by the distance in feet in which the blow is arrested, the result would be the measure in pounds of the impact, force of percussion, or blow. When a miner strikes a blow with a hand-hammer on the end of a drill-steel in drilling a "down" hole, two forces are acting; the muscular force in the effort to increase the velocity, and the force of gravity. At the moment the hammer strikes the end of the drill-steel, a velocity of from

20 to 40 ft. per second may be realized, and the effect is the same as though this maximum velocity, at the moment of impact, had been maintained throughout the whole stroke of the hammer. If the velocity at the moment of impact be assumed at 30 ft. per second, the force will be the same as if the hammer had fallen from a height expressed by  $\frac{V^2}{2g}$ , or the square of the velocity, divided by twice the constant for gravity; or  $\frac{900}{64.33} = 14$  ft. With a hammer weighing 4 lb. the stored-up energy will be  $14 \times 4 = 56$  foot-pounds. Assuming that the cutting-edge of the drill penetrates  $\frac{1}{16}$  in. into the rock before the energy of the blow is fully absorbed, then are 56 foot-pounds of energy being exerted through a distance of 0.0625 ft. Dividing 56 by 0.0625 gives 10,769 lb., the direct pressure which would produce the same effect as the miner striking with a 4-lb. hammer.

As the pressure is a maximum when the hammer first comes in contact with the drill-steel, and reaches a minimum when the penetration of the steel into the rock ceases, 10,769 lb., as found above, would more correctly represent the average pressure throughout the time of the blow.

As the Box drill is driven by an electric motor, it becomes a simple matter to measure the energy delivered to the hammer of the drill. First, the drill-mechanism was run without the hammer, and the current measured to determine the work due to friction within the drill; then the current was measured with the hammer drilling into rock under usual conditions. The difference in these two measurements, neglecting the slight friction of the hammer and the drill-steel, gives the measure of energy exerted in the hammer-blow and amounts to 6 amperes at 110 volts, or 29,195 foot-pounds. As the Box drill will penetrate hard granite at the rate of 3 in. per minute, with a drill-steel of 2.25 in. in diameter, the energy of the blow will be 116,780 lb., or over 58 tons, of static pressure. The hammer weighs 10 lb., has a 4-in. stroke, and strikes 600 blows per minute.

*Power Required.*—In drilling a hole 2.25 in. in diameter in hard granite from Platte Cañon, Colo., at a speed of 3 in. per minute, the Box electric drill required an electric current of

12 amperes at 110 volts, equivalent to 1.32 kw., or about 1.77 h.p. This means that 11.92 cu. in. per minute of granite chips have been removed with an expenditure of 1.77 h.p., or about 6.74 cu. in. of material per horse power per minute. A 3-in. air-drill on the same rock, and drilling at the same speed, will require at least 23 h.p., when drilling at an altitude of 10,000 ft. above sea-level. This gives only 0.518 cu. in. of material cut by the air-drill per horse power per minute; or 1 h.p. expended on the Box electric drill will cut 13 times as much material as the same amount of power expended on a 3-in. air-drill.

Under the average conditions that exist in the Rocky Mountain section, the power required by the Box electric drill in drilling a hole 2.25 in. in diameter at the speed mentioned above, varies from 1 to 1.5 horse power.

*Field of Operation.*—The Box electric drill was developed to meet the severe service in mining-operations, and after repeated trials under trying conditions it has shown that it will stand the test. There is a certain amount of prejudice to overcome, as is the case in any marked change in methods, but as the same amount of work may be accomplished in the same time, with one-tenth the power used in former methods, the electric system is bound to be recognized by progressive operators, and in good time it will have universal application.

Considering the many advantages in the installation of electric drills for excavations, quarries, sewers, subways and railroad construction, and the economy to be realized in the electrical transmission of power by the reduction in size of plant and facility in changing the position of drills to accommodate the work, the future for this type of drill is very promising.

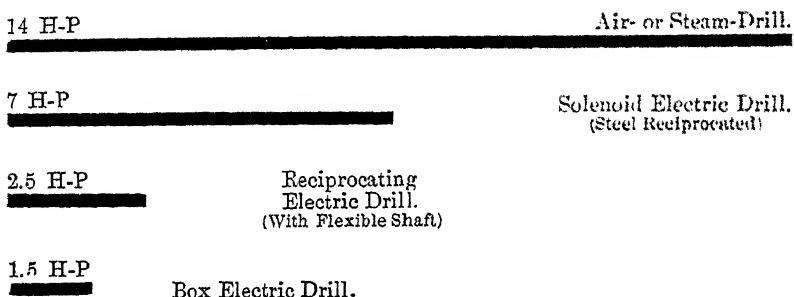
The remarkable results attained in the methods of long-distance electrical transmission of power, which renders available the great power-sources of water, or of coal- and oil-fields; the high standards in the manufacture of modern electric generators, motors and appliances; the increasing knowledge among operators concerning the care and control of electric apparatus; the desire for more economical methods in the work of excavation and mining;—all present most favorable conditions for the success of electric rock-drilling.

## COMPARISON WITH OTHER DRILLS.

*Relative Advantages.*—The advantages of an electric drill over one operated by steam, or air, may be appreciated when attention is paid to the great loss by condensation, radiation, or from leaky joints of pipes, as well as the heavy and expensive pipes and fittings necessary in the steam- or air-plants: whereas, the insulated electric cable may be very readily and cheaply installed.

The relative power required to operate the several types of drills is shown graphically as follows:

FIG. 8.



THE RELATIVE AMOUNTS OF POWER REQUIRED TO OPERATE DIFFERENT TYPES OF ROCK-DRILLS.

The objection is sometimes raised to the electric drill in that it provides no means for ventilating the workings in which the drill is operating. In reply it may be said that a perfect system of exhaust- or pressure-ventilation with fans can be designed, requiring very little power as compared with the enormous waste from compressed-air systems in which it is the custom at times to open wide a hose 1 in. or 1.25 in. in diameter. Furthermore, the liberation of compressed air from the drill, or the air-pipe, forces the foul air from the face of the workings back through the other workings of the mine, and furnishes an atmosphere for the miners to breathe, which is vitiated by atomized lubricating oil. Surely such air is not health-giving.

The electric drill does not vitiate the air; and, in addition, its use furnishes a satisfactory system of electric lighting, for the reason that incandescent lamps may be run from the drill-circuit.

The loss in efficiency of air-compressors at high altitudes is another strong argument in favor of the electric drill. The loss for air-compressors delivering air at 100 lb. gauge-pressure, at the different altitudes, compared with the same compressor at sea-level, is as follows:—

Altitude (Feet), . . . .	4,000	6,000	8,000	10,000	12,000
Loss of capacity (Per cent.), . .	22	28	33	39	44

There are no such corresponding losses in using the electric drill.

The hammer type of drill possesses a great advantage over the reciprocating forms, from the great saving in power due to the blow being transmitted through the drill-steel in the hammer-drill, instead of moving the drill-steel back and forth in the hole, as is the case in reciprocating-drills. A drill-steel 6 ft. long, weighing about 18 lb., used in connection with a 3-in. air-drill, with a stroke of 6.5 in., at the rate of from 300 to 400 blows per minute, requires at least 20 h.p. at sea-level, 22 h.p. at 5,000 ft. altitude, and 24 h.p. at 10,000 ft. altitude. The forward stroke of the reciprocating-drill ends in a blow in the rock; the return stroke must end on some form of cushion. The use of spiral springs for this purpose is a great source of trouble on account of their breakage.

In the Box electric drill, only the hammer moves, the drill-steel remaining at the bottom of the hole, except for a slight rebound due to the impact on the rock. When a drill-steel is reciprocated in a hole, considerable friction is produced, and in addition, the gauge of the cutting-edges is rapidly worn down. This is a strong point, for, as the gauge is preserved for a longer period, the holes are more perfect and a marked saving is effected in the cost of sharpening the drill-steels.

Water-power at a considerable distance from the mine may be developed, thus saving the transportation to, and the maintenance of, machinery in high altitudes and remote places. The difference in price of fuel between the mine-site and the railroad may also warrant the building of the power-station on the railroad.

#### PRACTICAL RESULTS.

The following is a summary of reports from various mines having the Box electric drill in regular operation:—

1. *Record in Tunnel-Work in Boulder County, Colo.*—"One of the machines drilled 12 ft. and 6 in. of hole 2.25 in. in diameter, in hard, closed-grained granite, in just two hours. The machine was set up in the tunnel at a point about 1,700 ft. from the generator. While the machine was in operation the ammeter on the switchboard indicated that it was using about one horse power."

"Arrived at tunnel about 1.00 p.m. Unpacked drill and set up; started to drill at 4.05 p.m.; drilled till 6.30 p.m.; drilling 3 holes, each 5 ft. deep, tore down and blasted 7.00 p.m. Driving tunnel about 7 ft. by 8 ft."

The following statement refers to shaft sinking in country-rock at the Ophir Mine, Anaconda, Colo., Cripple Creek District:

"You ask me how often we are obliged to clean the drill. I do not think we have averaged more than once a week.

"We have never had a drill stick or bind in a hole, or fail to clear itself of waste, or be deflected from its course by change of strata. We can easily begin at 8 o'clock and complete a round of holes, nine in number, 4 ft. deep (each), by 2.30 or 3 p.m., giving ample time for removal of drill and loading and shooting at the four o'clock shift. For instance, we set up the drill at 8 o'clock this morning, I sat by and watched the man complete five 4-ft. holes by 11.30 a.m.

"Started setting up at 7.30 a.m.; knocked down at 3.00 p.m.; drilled 30 ft.

"The drills are doing very satisfactory work, cutting as rapidly as air- or steam-drills, and accomplishing the work at a much less expense and power."

3. *Record at the Tunnel of Imogene Basin Gold Mines Co., Ouray, Colo. Rock very Soft with Mud-Slips Running through the Seams, and very Hard to Mix with Water. Altitude 11,000 Ft. above Sea-Level.*—"Cleaned breast and set up 11 a.m., drilled till 12 noon, started 1 p.m., drilled till 3 p.m., drilling 4 holes, making 20 ft.

"Engine uses about 6 gal. of oil (kerosene) in two 8-hour shifts, running one drill and four lights in daytime, and one drill and 14 lights at night. Cleaned breast and set up 1.50 p.m., drilling 4 holes, making 16 ft. at 3.20 p.m."

4. *Record of Experimental Work with the Box Electric Drill.*—Table I. gives the record of a test of a Box electric drill,

installed by Mr. Frank Fletcher of Parral, Mexico, which was made on April 14, 1904. The rock used for the test was hard granite from Platte Cañon, Colorado. The diameter of the drill-hole was 2.25 inches. The electric-current used was 11 amperes at 110 volts, which is equivalent to 1.62 horse-power.

TABLE I.—*Record of Test of Box Electric Drill.*

	Time.	Distance Drilled.		Rate of Drill- ing.	Time for Changing Drill-Steels.
		From.	To.		
	H. M. S.	Feet. Inches.	Feet. Inches.		
Started drill.....	9 24	0 0	.....	.....	.....
Stopped drill.....	9 31	0 0	1 4	16 in. in 7 minutes.	1 minute.
Changed steel, using side-tilting device:					
Started drill.....	9 32	.....	.....	.....	.....
Stopped drill.....	9 37	1 4	2 6	14 in. in 5 minutes.	.....
Changed steel, using side-tilting device:					
Started drill.....	9 38	2 6	2 9.5	3.5 in. in 45 seconds.	1 minute.
(Hole through rock) stopped drill.	9 38 45	.....	.....	.....	.....

The total depth drilled in 14 min. and 45 sec. was 33.5 in., which is at the rate of practically 2.23 in. per minute.

The following list of plants at which the Box electric drill is installed may be considered representative for the various localities named:

Ophir Mine, Anaconda, and Imogene Basin Gold Mines Co., Ouray, Colo.

Keystone Bromide Mining Co., Tres Piedras, N. M.

Bagdad Chase Gold Mining Co., Camp Rochester, Cal.

Miami Mining Co., Concord, N. C.

Akrokerri Mines Co., Ltd., Ashanti, S. Africa.

Frank Fletcher, Baca Station, Durango, Mex.

Compania Minera El Banco y Annexas, Oaxaca, Mex.

Gualterio C. Palmer, Zacatecas, Mexico.

Takata & Co., Tokio, Japan.



## Copper-Ore and Garnet in Association.

BY WILLIAM P. BLAKE, F.G.S., DIRECTOR OF THE ARIZONA SCHOOL OF MINES.

(New York Meeting, October, 1903.)

THE mineral, garnet, is a common associate of copper-ore in the southwestern portion of the United States. This association may be observed on a large scale in southwestern Arizona, in southern New Mexico and in Sonora, Old Mexico, especially at El Cananea, and at Nacozari. At other localities in Mexico the general occurrence of garnet "as a contact metamorphic mineral" in the copper-deposits of the Cretaceous limestone is noted by José G. Aguilera.<sup>1</sup>

It is usually found at the contact of this limestone with diorites, granites, etc., at numerous mines. The same authority also cites the occurrence of garnet in metamorphosed Cretaceous limestone alongside of diorite dikes in association with iron-ore.

Other foreign examples are found in the Banat and Servia, at Rezbanya in Hungary, and Offenbanya in Transylvania, and at Bogoslovsk in the Urals. In New South Wales, at Broken Hill, the gangue in the Proprietary mine is chiefly garnet, with quartz, opal and rhodonite.<sup>2</sup>

Also, in a recent paper by George Smith, entitled, "The Garnet-Formations of the Chillagoe Copper-Field, North Queensland, Australia,"<sup>3</sup> it is shown that in the copper-field of Chillagoe, Australia, where there are many lode-formations, about one-half of the number are garnetiferous, and many of this garnet-type are associated with magnetite, which is regarded as of analogous origin. The two minerals are found merging imperceptibly into each other.

These examples suffice to show the very general and wide distribution of garnet as an associate of copper-ore.

In addition to the localities already mentioned as known in Arizona may be mentioned the copper-mines and occurrences

<sup>1</sup> *Trans.*, xxxii., 500.

<sup>2</sup> Lindgren, *Trans.*, xxx., 611.

<sup>3</sup> See p. 467.

in the Dragoon mountains and in the Santa Rita range south of Tucson. Also in the Tucson mountains; the Sierritas, the Santa Catalina mountains and the copper-mines of the Lower San Pedro near Dudleyville. Two more important and instructive examples are found in the mines of the Twin Buttes Co. south of Tucson, and in the property of the Imperial Copper Co., in the Silver Bell district west of Tucson. It also occurs at the Atlas property in the northern continuation of the Imperial mine.

These and other examples which might be added are sufficient to show that in Arizona, as elsewhere, it is the rule, with scarcely an exception, that wherever either a plutonic ejection traverses limestone, or follows a plane of contact of the limestone and another rock and has the conditions requisite for metamorphic action, then garnet, and sometimes other silicates, are developed, and are found with the copper-ore. So, also, is garnet found in reefs or beds at and near the plane of contact of limestone with a granitoid rock. This reef-formation may be seen to an unusual extent at the Twin Buttes property, where there is a series of garnet-reefs in parallel lines along and near the contact with the granite, which appears to be eruptive. These reefs are copper-bearing and have the semblance of regular lodes. One evidence of the igneous intrusive character of the granite is the alteration of the blue, or dark-colored, limestone to white crystalline limestone near the contact.

Both at the Twin Buttes mine and the Imperial mines, where the garnet-formation is much more massive, copper-ore in the form of sulphide, usually chalcopyrite, is disseminated in the midst of the garnet in the form of grains, stringers or nodular masses. Such masses are found completely enveloped in garnet and are evidently of contemporaneous origin. Large blocks of massive garnet-rock quarried at the Twin Buttes often disclose, on breaking, nodules of copper sulphide so completely enclosed in a dense, granular, garnet-magma as to be fully protected from atmospheric agencies, remaining unchanged, and even unoxidized, in the croppings exposed to the weather. The occurrence of copper sulphides in a more or less vein-like and tabular form is also to be noted as indicative of deposition in open spaces. This form of occurrence may

be seen in the huge garnet-croppings on the Silver Bell (Imperial) and Atlas ground, particularly where quartz vein-stone is largely developed, the garnet itself having a tabular, vein-like structure, or crustification. In such cases the garnet cannot be regarded as other than a true vein-stone or gangue, and the garnet-reefs as having the nature of lodes,—a conclusion which is sustained when, as is often the case, vein-like inclusions of free quartz are found parallel to the general course of the main reefs of garnet-rock, and in the midst of the garnet.

The garnet found in association with limestone is usually the variety known as grossularite, the calcium-aluminum garnet, without iron enough to give it much color, but the wine-yellow and cinnamon-brown colors are common.

At the Cananea copper-mines in Sonora, Mex., there is a fine example of the conversion of a considerable area of limestone into garnet-rock without the complete obliteration of the original bedding-planes, or partings, of the strata. This may be seen between the croppings of the Veta Grande and the Democrita mines at the place where, according to the plicated structure of the region, croppings of limestone would be expected to occur. Instead of limestone, massive garnet-rock is found, with here and there bunches of included copper sulphides. These croppings afford a good example of areal metamorphism in contrast with local linear metamorphism following planes of fracture, or of contact.

In the case of reefs of garnet-rock in Arizona, the croppings are usually rough and irregular, though preserving in the main a distinct tabular form. The garnet being much less soluble than the country-rock of limestone, the croppings are usually found in bold relief above the general surface, and can be readily traced.

It is usual to regard these garnet-formations as the result of contact metamorphism; the alteration consisting of a combination of elements, or compounds, pre-existing in the strata of limestone, such as siliceous and aluminous layers,—a condition so well exemplified by the metamorphic limestone of Sussex county, N. J., and of Essex county, N. Y., where bands of silicates and oxides shown by layers of such crystalline minerals as augite, chondrodite, garnet, spinel and corundum, are common.

The conditions and phenomena of the copper-garnet formations of the southwest, herein but partially set forth, indicate a somewhat different origin. They support and justify the conclusion, as regards the silica, that the formations are largely of extraneous origin, due to siliceous emanations traversing limestone, following, in general, contact-planes, or planes of fracture, or fissures. These siliceous emanations supplying the silica for combination with the lime and alumina of the beds, have resulted in the formation of veins and beds of garnet-rock, and frequently of other silicates, especially epidote, a common associate, also, of copper in southwest Arizona and elsewhere. The general presence of copper sulphides in these garnet-reefs, together with an excess of silica, shown by the presence of quartz, gives the garnet-reefs a lode-like character, the garnet becoming a true veinstone, or gangue, for the copper-ore.

I am not aware that there has been any direct statement of this view of the origin of garnet-rock, and of the recognition of linear reefs, or formations of garnet, as true lodes or veins which may be followed and exploited as veins of quartz are followed.

In the paper of Lindgren, before cited, reference is made to certain unpublished observations by S. F. Emmons, "that garnets replace limestone at Clifton, Arizona, apparently as part of the phenomena of mineralization."

Aguilera also writes<sup>4</sup> in regard to the copper-deposits of Mexico, "The gangue in these contact-viens is quartz; the ores are chalcocite, chalcopyrite and bornite. These are generally accompanied by grossularite, iron-garnet, wollastonite, tremolite and vesuvianite, all of which are products of alteration formed at the expense of the Cretaceous limestone through the agency of heated waters."

In the description of the garnet-formations of the Chillagoe copper-field by Mr. Geo. Smith,<sup>5</sup> the garnet is regarded as intrusive and of igneous origin. After noting the absence of signs of replacement of the limestone, and describing a mass of garnet 200 ft. thick enclosed on either side by granite, Mr. Smith says: "The close association of the garnet with the limestone apparently points to an essential connection between them, yet the former rock (garnet), occurring elsewhere and

<sup>4</sup> *Trans.*, xxxii., 510.

<sup>5</sup> See p. 473.

presumably without such association, has been unhesitatingly classified by eminent authorities as of igneous origin."

The chief point which it is desired to present and to emphasize in this paper is that we are justified in regarding many copper-bearing garnetiferous reefs as true lodes originating by the efflux of silica, or its passage through fissures, entering into combination with the bases of the strata, such as lime, alumina and iron-oxide, crystallizing as garnet or other silicates and assuming structural characters, as gangue or veinstone, analogously to quartz lodes; and that many such reefs, consisting chiefly of garnet, are not the result of combination of indigenous silica and basic substances in the strata.

From an economic point of view, garnetiferous ores present some difficulties in working. The high specific gravity of garnet prevents its elimination by concentration-processes. In smelting, the garnet forms a pasty slag, and is a worse than useless burden in the furnace. The true direction in which to work such ores appears to be by lixiviation. By crushing and roasting, the copper becomes soluble, while the garnet gangue is not. It is quite probable, also, in some sections of the country, at least, that the lixiviated garnet can be utilized as an abrasive.

## The Refining of the Precipitates Obtained by Means of Zinc in the Cyanide Process of Gold and Silver Extraction.\*

BY G. HOWELL CLEVINGER, B.S., A.M., DEADWOOD, SO. DAK.

(New York Meeting, October, 1903.)

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### INTRODUCTION.

ATTEMPTS to discover, for the cyanide process, a better precipitant than zinc have been unsuccessful. Hydrogen sulphide, sulphurous acid, ferrous sulphate, etc., precipitate gold and silver from cyanide solutions either not all or incompletely.

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\* This investigation was made in the metallurgical laboratories of Columbia University, New York City, under the direction of Prof. Henry M. Howe, and its details and results were fully stated in a thesis, of which this paper gives the substance, together with some additional and more recent material.

Thanks are due to Professors H. M. Howe and Edmund H. Miller, and Mr. Bradley Stoughton, of the above University; Messrs. C. W. Merrill and Allan J. Clark, of Lead, S. D.; and Mr. J. V. N. Dorr, Terry, S. D., for aid given in the preparation of this paper.

Aluminum has been tried, but it costs too much, and cannot be used in solutions containing lime. Charcoal has been used to a small extent, in treating acid tailings, the solutions of which were acid or very weakly alkaline. It involves an excessive cyanide-consumption; must be used in large bulk; and subsequently all burned to ash, with risk of losing gold and silver. On the other hand, the ash is easily melted, almost fluxing itself; and the bullion is fine.

Of the many proposed electrical methods of precipitation, only the Siemens-Halske or some modification of it has proved successful. But this method is losing rather than gaining ground. Mr. John Yates, in a paper<sup>1</sup> comparing the zinc and the electrical methods of precipitation, shows that the former presents decided advantages. In fact, zinc may be said to take the lead more and more, as its use is better understood.

One of the weak points in this method has been the refining of the zinc-precipitates (containing from 4 to 50 per cent. of bullion, and from 20 to 40 per cent. of zinc, together with base metals, metalloids, silica, lime, etc.), from which it is difficult to obtain clean bullion without loss.

Zinc is used as a precipitant in two forms;—shavings or filiform zinc, and zinc-dust. The latter, first proposed, I think in connection with the bromo-cyanide process of Sulman and Teed,<sup>2</sup> did not win much favor until it was adopted by C. W. Merrill, at Marysville, Montana, and later by the Golden Gate mill at Mercur, Utah.<sup>3</sup> Lately, a number of large American works have adopted it. Its general use is hindered by the difficulty of getting zinc-dust of uniform quality and free from oxide, and by the circumstance that, for some unexplained reason, zinc-dust precipitates do not yield as readily to acid-treatment as do those from zinc-shavings.

### I. NATURE OF PRECIPITATES.

Whether from zinc-shavings or dust, the precipitates are apparently alike in general composition. Mr. Willam Magenau gives analyses of both, which support this statement.

<sup>1</sup> The Cyanide Process, *Proceedings of the Chemical and Metallurgical Society of South Africa*, Sept. 19, 1896, vol. i., p. 254.

<sup>2</sup> *Journal of the Society of Chemical Industry*, Dec. 31, 1897, pp. 961-970.

<sup>3</sup> Some Observations on the Practice of the Cyanide Process at Mercur, Utah, *Mining and Scientific Press*, San Francisco, May 5, 1900, p. 492.

As to the forms or combinations of the various constituents, it has been generally assumed by the authorities that the metals, gold, silver, copper, etc., are present in the metallic state. A few writers have suggested that the black precipitate may be an intimate combination of gold with something else. Gage<sup>4</sup> thinks it is a double cyanide of gold and zinc. Rose<sup>5</sup> says:

"Pure zinc has only a slower action on solutions of potassium cyanide than ordinary commercial zinc containing about one per cent. of lead; and the action of the gold-zinc couple, formed by the black deposit of gold (which may be really a compound of gold and zinc) and the unaltered zinc is still more vigorous."

The following are the conditions of practice under which gold and silver are precipitated by zinc from cyanide solutions:

In a slightly alkaline cyanide solution, containing usually less than 1 oz. of gold and silver per ton (2,000 lb.) of solution, and in many cases as low as from 0.05 to 0.02 oz. (and even lower in the case of final washes), or 1 part of gold and silver in from 24,000 to 1,200,000 parts of solution, the gold and silver are precipitated by the strongly positive metal, zinc, which is in a finely divided state and hence very active chemically.

F. Mylius and O. Fromm<sup>6</sup> have made an interesting series of experiments upon very dilute solutions of the salts of different metals,—silver, gold, copper, etc. Upon immersing a zinc rod in the solutions of these metals they found that the metal in solution was precipitated as a spongy black mass, which they determined to be a definite alloy of zinc and the metal. As a result of their investigation they concluded (among other things) that the metals are capable in the moment of their separation, and at ordinary temperatures, of combining with each other; and that, by the action of positive metals upon dilute solutions of negative metals, alloys are electrolytically formed.

In the case of the silver-zinc alloy, it was found that if the strength of solution went above 1 part of silver in about 33,000 parts of solution, white crystalline metallic silver was precipitated. If less silver was in solution than 1 part in 33,000, the black precipitate or alloy of silver and zinc was formed.

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<sup>4</sup> *Practical Cyanide Operations*, 1st ed., p. 105.

<sup>5</sup> *Metallurgy of Gold*, 3d ed., p. 362.

<sup>6</sup> *Berichte der Deutsche Chemische Gesellschaft*, 1894, Part I., p. 630.



The degree of dilution necessary to form these alloys probably differs with the different metals; and it also seems possible that there is a whole series of these alloys formed by the action of zinc in solutions of varying metal-content. All the experiments of Mylius and Fromm were made with neutral or slightly acid solutions.

Are such alloys formed when gold and silver are precipitated from a cyanide solution by zinc? In that case, the solution is slightly alkaline, and the proportion of metal is about from 1 : 24,000 to 1 : 1,200,000.

It is well known that gold and silver precipitated from a cyanide solution (not containing too much of these metals) come down as the black flocculent precipitate, looking like that obtained by Mylius and Fromm.

This precipitate, after very fine sifting, which undoubtedly removes most of the free zinc, still contains a high percentage of zinc, although but little can be seen by the microscope. Moreover, when the metal-content of the solution is above a certain point, the zinc assumes the color of gold or silver, as the case may be; and the gold or silver is precipitated in the metallic state, or a near approach to it.

Philip Argall<sup>7</sup> says:

“When the solutions pass 1.5 oz. per ton the gold on the shavings assumes a yellow color, and passing 2 oz. and upward is usually golden yellow in the first compartment, shading off in the following compartments to the usual black-colored deposit that collects on the zinc in ordinary precipitation from poor solutions.”

This passage describes in fact the appearance of the gold precipitated by zinc from solutions of varying gold-content, the other conditions remaining almost the same. The solution upon entering the precipitation-box contains its maximum gold-content (in the above case, 2 oz., and upwards), but as it progresses through the several compartments of the precipitation-box, and the gold is precipitated, the solution becomes poorer and poorer in gold, and the character of the precipitate is changed. In the first compartment the gold is probably in the metallic state or a near approach to it. In the second is a compound containing more zinc, and so on throughout the

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<sup>7</sup> Cyaniding Telluride Ores, *Mineral Industry*, vol. vi., p. 379.

series until we finally reach an alloy, containing the maximum percentage of zinc.

## II. EXPERIMENTAL WORK UPON ALLOYS.

In order to investigate the formation of zinc-gold, and zinc-silver alloys from cyanide solutions, a simple apparatus was fitted up, consisting of two ordinary oil-barrels, as tanks, placed with the top of the second lower than the bottom of the first, so that a current of solution could be passed from one to the other through a series of four interposed glass bottles, containing the zinc precipitant. The shavings having been placed in the bottles, the silver- or gold-solutions were caused to pass very slowly through them.

*Silver.*—The first experiments were made with 400-lb. samples of the following solutions:

	Ag. Ounces Per Ton.	KCN. Per Cent.	Pounds Per Ton.	NaOH. Pounds Per Ton.
I., . .	1.28	0.18	3.6	1
II., . .	2.06	0.19	3.8	1
III., . .	0.68	0.20	4.0	none

In each case, the silver was deposited in a hard, thin, adherent coating, which could not be removed for separate analysis. It varied in color from light straw-yellow to copper-red, except that, in case I., it was from gray to black in spots where the shavings were somewhat rough. In cases I. and II., a considerable quantity of hydrogen (recognized by collecting and burning it) was evolved. In III., there was very little hydrogen, and the action generally was much slower. As already observed, the precipitate could not be removed for examination or analysis. But I have found an alloy containing from 40 to 50 per cent. of zinc and the balance of silver to show much the same characteristics. The rough surface of this alloy has the same copper-red color, and even the polished surface has a very distinct reddish tinge. With acetic acid, little or no action takes place in either case. Other reagents, such as chromic acid, hydrogen peroxide, etc., give the same reactions in both cases.

That this coating is more energetic than zinc as a precipitant of gold, copper and silver, may be seen by watching the precipitating-bottles. At first the precipitation is very

slow; but after the shavings have assumed the copper-red hue it goes on much more rapidly.

*Gold.*—A similar experiment was tried with a cyanide solution containing 0.2 per cent. or 4 lb. KCN per ton, 0.25 oz. gold and 1 lb. NaOH per ton. When this solution was passed through them, the zinc shavings rapidly assumed a dark purple-black color, which deepened as the process went on, until, as the precipitate became heavier, it detached itself from the zinc and fell to the bottom of the bottle.

To avoid oxidation, this fine precipitate was dried in an atmosphere of illuminating gas, after which it was shaken lightly on a 100-mesh screen to remove particles of metallic zinc. The resulting product was found to contain 40.11 per cent. of zinc, the rest being probably gold. It was found to oxidize very rapidly in the air. I have noticed the same phenomenon in commercial practice. When zinc shavings pretty well charged with precipitate are removed from the zinc-box and exposed to the air, they become quite warm through rapid oxidation.

For purposes of comparison, an alloy of gold and zinc, containing from 40 to 50 per cent. of zinc and the rest gold, was prepared in the ordinary way. It formed a hard white mass, rapidly acted upon by nitric acid; metallic gold being set free. In a finely divided state, it is also a very energetic precipitant of gold.

The purple-black precipitate, when subjected to pressure under an agate pestle, assumed the same appearance; and it acted in the same way with nitric acid and other reagents.

A portion of the precipitate allowed to remain in the precipitation-bottles after all of the free zinc had been consumed, still continued to precipitate gold with evolution of hydrogen. At first the action was very energetic; but as the zinc in the precipitate decreased, it became less and less active until it apparently ceased altogether.

The purple precipitate first shaded off into a dark brown, then through various shades of yellow until it assumed a metallic luster and a golden color. In this last stage it does not seem to contain much zinc; the exact amount, I have been unable to determine.

*Conclusions.*—It may be inferred from these experiments:

(1). That the precipitates formed by zinc in the cyanide pro-

cess are probably alloys or intimate mixtures of zinc and the metals in solution; and

(2). That they follow certain regular laws as to their formation and composition.

### III. PRESENT METHODS OF REFINING.

The various metals and elements may be considered as occurring in the following combinations: gold, silver and copper, as intimate mixtures or alloys with zinc; mercury, as an amalgam, probably of zinc; lead, arsenic and antimony, in unknown combinations; iron and aluminum, as hydroxide, oxide or silicate, due to clay; calcium, as sulphate, carbonate or hydroxide; magnesium, probably as carbonate; silicon, as silica (sand) and silicates; zinc, (*a*) in metallic form, due to excess of zinc used in precipitation, (*b*) as oxide, due to oxidation during handling of the precipitates, and (*c*) in combination with gold, silver, copper, etc., as intimate mixtures or alloys. Cadmium, resembling zinc in its chemical behavior and being a strongly positive metal, probably forms similar alloys when present in the zinc used for precipitation.

Finally, there are in the precipitates complex organic compounds, of doubtful nature. The numerous impurities mentioned above, coming from the ore, the zinc (which is, in practice, never pure), and the lime used for neutralizing acidity of ore, render the refining difficult.

There are three general methods of refining: (1) roasting, with or without an oxidizing agent (such as niter), then smelting oxidized precipitates with suitable fluxes in graphite crucibles; (2) acid-refining, which consists of preliminary treatment with some acid, usually sulphuric acid, then smelting acid-treated precipitates with proper fluxes in graphite crucibles; and (3) smelting with lead, either with or without preliminary acid-treatment.

#### 1. *Refining by Roasting.*

This method has been more or less used since the introduction of the cyanide process. At one time it was the general practice in South Africa; but it has never prevailed in the United States.

The roasting of the precipitates is usually done in cast-iron muffles, heated with coal or wood. The precipitates are spread

out in thin layers upon large iron trays, and stirred from time to time during the roasting. Roasting in pans, heated from the bottom by an open furnace, is not considered as good practice as roasting in a muffle. In many cases, oxidation is aided by the use of niter, applied to the precipitates as a strong solution or mixed with them in the dry state; from 3 to 5 per cent. is the usual amount used; any excess over the quantity necessary to oxidize the base metals present is to be avoided, as it rapidly destroys the graphite crucible used for melting. The oxidized precipitates are mixed with suitable fluxes and melted in graphite crucibles.

The main object is a thorough oxidation of all base metals, so that in the subsequent melting they may be carried into the slag.

The advantages of this method are,—the simplicity of operation and the requirement of no acids or other chemicals aside from fluxes.

Its disadvantages are,—the production of low-grade bullion; the heavy losses of precious metal; the difficulty of roasting properly; and the rapid corrosion by the strongly oxidizing charge of the graphite crucibles used for melting,—especially when niter is employed.

These drawbacks are so serious that the method is rapidly going out of use.

## *2. Refining with Acid.*

Sulphuric acid only has been used to any great extent for the preliminary treatment of the precipitates. Nitric acid has not proved satisfactory. Hydrochloric acid seems to give as good results as sulphuric, but costs more, and might possibly occasion the evolution of chlorine, which would dissolve gold. Acetic acid has been suggested, but I have found no record of its use in practice.

The acid-treatment is usually carried on in wooden tanks, sometimes lined with sheet-lead. At some plants, the mixed precipitates and acid are heated in the tank by steam introduced through the mass near the bottom. At some of the older works this heating is done in a large leaden dish supported over a furnace. In many places, however, the heat generated by the mixture of the concentrated acid and water

is deemed sufficient. Mechanical stirrers are used, but in many cases the mixture is stirred by hand with a wooden paddle.

The strength of acid varies greatly. In general it is safe to say that the best results are obtained by using rather dilute acid (1 part of concentrated acid to from 8 to 10 parts of water), at least for the first treatment. If the acid is too strong, there is greater danger of the formation of the less soluble base-metal salts, which cannot be completely removed by washing; this largely defeats the removal of zinc, which is the chief purpose of the acid-treatment.

In small and poorly-equipped plants, the precipitates are allowed to settle and the clear solution siphoned off, wash-water is then added and the mass again allowed to settle and the clear solution siphoned off. This is repeated until the precipitates have been completely washed. This procedure takes much time; zinc sulphate can never be completely removed in this way; and serious losses may arise from fine particles of precipitate going over with the wash-water.

In the best practice a vacuum-filter or a filter-press is used. By the latter method a rapid and complete separation of the solid matter from the liquid may be made with but very little loss.

After the zinc is in solution, the important point is the complete removal of zinc sulphate by thorough washing. The thoroughly-washed precipitates are best dried in a cast-iron muffle heated from the top only, which prevents loss through boiling and spattering. (In some cases a roast is given at this point.) The material is then mixed with fluxes (borax, soda, sand, fluorspar, etc.) in such proportions as to give, with all of the impurities and base metals, a fluid slag that does not contain shots of bullion. The melting is done in graphite crucibles, as in refining by roasting. C. Butters<sup>8</sup> has used a reverberatory furnace for this purpose.

The advantages of acid-refining are, that it produces fairly fine bullion (the fineness usually depending upon the care exercised in conducting the operation), and that the losses are not so heavy as in the roasting process. Its disadvantages are,

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<sup>8</sup> *Journal of the Chemical and Metallurgical Society of South Africa*, vol. iii., No. 3, July, 1902, p. 23.

that it calls for a considerable equipment and the losses due to handling are comparatively large.

With careful work the loss is usually said to be about 1 per cent. of the value, but there is little doubt that, in most cases, it is larger. Mr. C. W. Merrill<sup>9</sup> states the losses in ordinary methods of refining (meaning, probably acid-refining) at from 2 to 6 per cent. Alfred James<sup>10</sup> gives the losses by the roasting method as from 0.5 to 6 per cent., and adds:

“Acid treatment yielded results varying with the kind of acid used, nitric acid showing a greater loss than hydrochloric and sulphuric. On the other hand, with impure zinc, that in ordinary use, nitric acid gave the purest bullion, but with lead-free zinc, dilute sulphuric acid gave both the purest bullion and the lowest acid loss.”

One trouble in acid-refining is the formation of poisonous gases when the acid attacks the precipitate. Any cyanide remaining in the latter will thus form the deadly hydrocyanic acid. Arsenic (which may have been in the ore or in the zinc) forms hydrogen arsenide, which is fatal if inhaled. Several men have lost their lives by breathing hydrogen arsenide at the Golden Gate mill, Mercur, Utah, where ores containing arsenic are treated. The danger was finally overcome at this mill by using nitric acid in connection with the sulphuric acid (about one part of nitric to two of sulphuric), and ventilating thoroughly with exhaust-fans. The nitric acid oxidizes hydrocyanic acid to harmless cyanic acid, and the hydrogen arsenide to arsenic acid. Moreover, it aids in the solution of zinc, which is much more soluble in sulphuric acid in the presence of an oxidizing agent.

In some plants the precipitates are roasted before being treated with acid. This generally results in the more complete removal of zinc and other impurities, but the expense and probable loss involved in this preliminary step make its general employment scarcely advisable. As already remarked, precipitates formed by zinc dust are less completely acted upon by acid than those formed by zinc shavings. An instance in point came to my notice in the Black Hills, at a mill using zinc dust. When dilute acid was added to the precipitates in the acid-tank, little if any action could be observed. The mixture was left in the tank for a week, being agitated and warmed

<sup>9</sup> Paper read before The Black Hills Mining Association.

<sup>10</sup> *Transactions of the Institution of Mining and Metallurgy*, London, 1897-1898, p. 5.

from time to time by blowing steam through the mass. At the end of the week, it was filtered, and found to contain almost as much zinc as at the start. This experience led to the substitution of zinc shavings at the mill in question.

At the Golden Gate mill<sup>11</sup> at Mercur, Utah, on the other hand, zinc dust is used as a precipitant, and no such trouble is encountered with the acid-treatment. The preliminary roast, and the use of nitric acid, already mentioned, probably solve the problem in this case. It would be interesting to know what percentage of zinc remains in the acid-treated material.

At the Homestake Mining Co., Lead, S. D., zinc dust<sup>12</sup> is likewise used as a precipitant; but although the acid-treatment is thorough, it leaves in the precipitates from 4 to 12 per cent. of zinc.

Several causes may contribute to this partial failure of the acid to dissolve the zinc. As already observed, the remaining zinc may be due to the incomplete washing out of zinc sulphate; but besides the sulphate there is always more or less zinc remaining unattacked by the acid, whether dust or shavings have been used. This is especially true when sulphuric acid is employed.

The causes of this phenomenon seem to be:

1. The conditions of precipitation (strength of solution, metal-content, etc.) may be such as to form zinc alloys not readily attacked by the acid.

2. All commercial zinc contains more or less lead; and in some cases, lead is added intentionally (by immersing the shavings in a dilute solution of lead acetate) in order to promote precipitation by forming the lead-zinc couple. All the precipitates, therefore, contain more or less lead; and when they are treated with sulphuric acid the resulting insoluble lead sulphate tends to form a protective coating over the unattacked material, and thus prevent further solution of zinc and other impurities. Moreover, calcium is always present, and calcium sulphate has a similar tendency.<sup>13</sup>

<sup>11</sup> Private notes of Mr. Bradley Stoughton.

<sup>12</sup> Private Communication.—G. H. C.

<sup>13</sup> At the Homestake mill, a preliminary treatment with very dilute hydrochloric acid dissolves the lime, attacking the zinc but slightly. The treatment with sulphuric acid, for the removal of zinc, etc., follows.



This residual zinc is certainly a disadvantage in the subsequent melting. Slags containing zinc usually carry high values in gold and silver; and, besides, more or less zinc finds its way into the bullion.

Mr. A. Whithy<sup>14</sup> gives an analysis of acid-treated and calcined precipitates, said to illustrate the product of the Rand, S. A. R., which shows 7 per cent. of zinc oxide.

I have recently found 2.29 per cent. of zinc in a sample of acid-treated precipitates from a mill in the Black Hills, where the treatment is very carefully conducted.

### 3. *Smelting with Lead.*

A great many of the early cyanide plants in the United States shipped their precipitates to smelters.

The method of refining at the Balbach Smelting and Refining Co.'s plant at Newark, N. J., is briefly outlined as follows:

The precipitates, tied up in paper sacks in parcels of from 1 to 5 lb., are charged from time to time upon the bath of molten lead in a cupelling-furnace. The gold and silver are quickly absorbed by the lead, and until the mass is well melted, the precaution is taken to keep all the drafts closed. Subsequently, the slag is removed by skimming.

After the cupellation the bullion is passed through the regular process of refining.

Sometimes the material is briquetted, and sometimes it receives an acid-treatment before cupellation; but this is not generally done, since the precipitates are charged in small parcels and the small amount of zinc introduced in this way does not materially affect the cupellation.

Smelting with lead has not been adopted by cyanide plants as a means of refining their product until quite recently. A plant in South Africa, which had been experimenting with it for some time, has lately adopted it. Mr. C. W. Merrill,<sup>15</sup> metallurgist for the Homestake Mining Co., in 1900 worked out a system of refining in which smelting with lead was the prominent feature.

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<sup>14</sup> *Journal of the Chemical and Metallurgical Society of South Africa*, vol. iii., No. 3, July, 1902, p. 21.

<sup>15</sup> Paper read before the Black Hills Mining Association.

*Tavener's Method.*—Mr. P. S. Tavener<sup>16</sup> describes the method employed by him at the Bonanza Limited, S. A. R., in August, 1899. This, I believe, is the only place in South Africa at which it has been used. He compares the operation to scorification on a large scale. It consists in smelting the fine precipitates and fine zinc in a reverberatory, with litharge, assay-slag, slag from former operations, silica and saw-dust (the latter stirred in with a rabble, after fusion, to reduce lead from the slag). The base bullion resulting from this operation is cupelled in a special cupelling-furnace.

Mr. Tavener claims that this process is less costly than acid-treatment; that it leaves no by-products (slags and litharge being worked over again); that it involves less loss in handling materials, and that it effects, in practice, a more complete extraction. It may be added, on the other hand, that his reverberatory is more expensive than ordinary melting-furnaces, and that the successful operation of the process requires considerable metallurgical skill.

*Merrill's Method.*—The method developed by Mr. Merrill at the Homestake plant, Lead, S. D.,<sup>17</sup> comprises a preliminary treatment with dilute hydrochloric acid; the removal of the liquid through a filter-press; a subsequent treatment with sulphuric acid; the washing and drying of the precipitates; mixing with litharge, borax, silica and powdered coke; sprinkling with lead acetate; and briquetting under a pressure of from 2 to 3 tons per sq. in. The briquettes are melted upon the hearth of an ordinary cupelling-furnace, the resultant slag is run off, air is turned on and the lead is cupelled in the same furnace. The resultant metal, 975 to 985 fine, is run into bars. Cupel-slags and bottoms are put through a blast-furnace, recovering the lead for the next cupellation; and the litharge is added to the next charge of precipitates. Mr. Merrill says that the blast-furnace slag carries less than \$5 per ton in value; that the total cost of refining is less than 0.75 per cent. of the values contained in the precipitates, and that the loss (less than 0.1 per cent.) goes to the next precipitate. The total loss in refining is given as less than 0.1 per cent.

<sup>16</sup> The Lead-Smelting of Gold-Slimes, *Journal of the Chemical and Metallurgical Society of South Africa*, vol. iii., No. 6, p. 70, October, 1902.

<sup>17</sup> Paper read before the Black Hills Mining Association.

*Comparison of Costs of Acid-Refining and Lead-Smelting.*

Table I. gives Mr. Tavener's comparison of costs, and bullion recovered by the two methods, as practiced at the Crown Deep, South Africa. The clean-up was halved, one-half being treated by the usual acid-refining and the other half by lead-smelting, as described by Mr. Tavener.

For convenience, I have reduced pounds sterling to dollars, in reproducing Mr. Tavener's figures.

TABLE I.—*Comparison of Costs of Acid-Refining and Smelting-Processes.*

Costs for Acid-Treating, Calcining and Smelting.		Lead-Smelting Method.	
721 lb. sulphuric acid.....	\$54.65	10 bags coal, \$1.80.....	\$18.00
Crucibles, clay liners, stores fluxes, etc.....	53.40		
Coal.....	13.74		
Total.....	\$121.79	Total.....	\$18.00
	Oz.		Oz.
Smelted gold.....	443.727	Gold recovered.....	552.452
Fine by-products.....	31.253	Value of slag not credited.....	
Gold recovered.....	474.980		
Cost \$0.256 per fine oz.....	\$0.256	Cost \$0.032 per fine oz.....	\$0.032

"In the above, labor costs are not included. Labor was less in the lead-smelting."

"The lead loss was estimated to be 12 per cent.; this, as well as the cost of material and labor for tests, is not given in the above, which will likely increase the cost of refining to from \$0.04 to \$0.05 per fine oz. of gold produced."

"In three comparative tests, made upon the same material, lead-smelting yielded from 10 to 11 per cent. more bullion than did acid-refining."

C. W. Merrill<sup>18</sup> gives the cost of refining at the Homestake as \$0.15 per fine oz. of bullion. Mr. Merrill has since furnished me with the very complete cost-sheet, given in Table II. It is unfortunate that the South African costs are not more fully given. A comparison would be most interesting.

TABLE II.—*Detailed Costs of Refining Cyanide Precipitates at Lead, So. Dak., Sept. 29, 1903.*

<i>Acid-Treatment.</i>			
Labor, . . . . .			\$14.00
Supplies, Muriatic acid, 435 lb. @ 0.03, .	\$13.05		
Sulphuric acid, 2,110 lb. @ 0.0175, .	36.92		
			<u>49.97</u>
			<u>\$63.97</u>

<sup>18</sup> The Metallurgy of Homestake Ore, see p. 596.

*Emptying Press and Sampling.*

Labor, . . . . .			\$5.50
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*Drying.*

Labor, . . . . .		\$9.00	
Supplies, Coal, 1 ton, . . . . .	\$3.50	3.50	
		<hr/>	12.50

*Fluxing.*

Labor, . . . . .		5.50	
Supplies, Borax, 575 lb. @ 0.089, . . . . .	51.18		
Coke, 115 lb. @ \$8.50 Tn., . . . . .	0.49		
Lead acetate, 70 lb. @ 0.089, . . . . .	6.23		
Litharge,* 502 lb. @ 0.05, . . . . .	25.10	83.00	
		<hr/>	88.50

\* Litharge is used repeatedly, losses being replaced by addition of pig-lead to cupel. The above charge is for the average monthly loss of lead.

*Sampling and Removing to Assay-Office.*

Labor (including hauling), . . . . .		8.00
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*Burning Press Cloths.*

Labor, . . . . .		7.50	
Supplies, Coal, 0.5 ton @ \$8, . . . . .	4.00	4.00	
		<hr/>	11.50

*Briquetting.*

Labor, . . . . .		11.25	
Supplies, Power, 60c. per hour, . . . . .	8.40		
Oil, . . . . .	0.21	8.61	
		<hr/>	19.86

*Drying Briquettes.*

Labor, . . . . .		6.00	
Supplies, Coal, 2 tons @ \$3.50, . . . . .	7.00		
Pans, . . . . .	1.50	8.50	
		<hr/>	14.50

*Preparing for Blast-Furnace Run.*

Labor (breaking old test; transferring material, etc.), . . . . .		10.12	10.12
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*Cupel-Furnace.*

Labor, . . . . .		32.50	
Supplies, Coal, 1.5 tons @ \$8, . . . . .	12.00		
Wood, 4 cords @ \$5, . . . . .	20.00		
Clay, 200 lb. @ 0.005, . . . . .	1.00		
Power (blower), . . . . .	1.50	34.50	
		<hr/>	67.00

*Refining and Shipping.*

Labor, . . . . .		5.50	
Supplies, Coke, . . . . .	2.13		
Crucibles, . . . . .	1.43		
Borax, . . . . .	0.89		
Canvas, . . . . .	0.30	4.75	
		<hr/>	10.25

*Blast-Furnace.*

Labor (running, cleaning crucible and re-assembling), . . . . .		13.75	
Supplies, Coke, 0.8 ton @ \$8.50, . . . . .	6.80		
Power (blower), . . . . .	0.75	7.55	
		<hr/>	21.30

<i>Sampling By-Products.</i>			
Labor, . . . . .		\$1.56	\$1.56
<i>Making New Test.</i>			
Labor, . . . . .		3.00	
Supplies, Cement, 475 lb. @ 0.011, . . . . .	\$5.23		
Ground limestone, 161 lb. @ 0.01, . . . . .	1.61	6.84	
			9.84
<i>Installing New Test.</i>			
Labor, . . . . .		3.00	
Supplies, Cement, 100 lb., . . . . .	1.10		
Limestone, 50 lb., . . . . .	0.50	1.60	
			4.60
<i>Re-Treating Matte.</i>			
Labor, . . . . .		3.00	
Supplies, Coke, . . . . .	0.50		
Wood, . . . . .	0.75		
Iron, . . . . .	1.00	2.25	
			5.25
<i>Grinding Fluxes.</i>			
Labor: on litharge, . . . . .		8.75	
lead acetate, . . . . .		1.25	
coke, . . . . .		1.25	
slags, . . . . .		0.62	
		11.87	
Supplies, Power, . . . . .	3.56	3.56	
			15.43
<i>Assays.</i>			
Labor, . . . . .		14.00	
Supplies, . . . . .	7.00	7.00	
			21.00
<i>Guards.</i>			
Labor, . . . . .			26.00
<i>Superintendence.</i>			
Labor, . . . . .			51.00
Total Labor, . . . . .		\$242.05	
Supplies, . . . . .		225.63	
			\$467.68

The figures of the cost of acid-refining, given in Table III., were obtained by me at the Horseshoe mill, Terry, S. D.

At this plant, clean-ups are made twice a month, which, together with the fact that the precipitation is carried on in 160 separate precipitation-boxes, makes the labor-item high.

The mill crushes wet, so that a large volume of solution is passed through the zinc in proportion to the tonnage of ore treated.

This seems to allow a large amount of lime, alumina, magnesia, etc., to separate in the zinc-boxes, greatly increasing the bulk of the precipitates and causing a high consumption of acid and fluxes.

The figures of cost include all items of expense from the time the washing of the zinc is commenced until the bullion is turned out in bars.

TABLE III.—*Cost of Acid-Refining at the Horseshoe Mill, Terry, S. D.\**

Labor, 10½ days @ \$4, . . . . .	\$42.00
10½ days @ \$3, . . . . .	31.50
Total labor, . . . . .	\$73.50
Supplies, 2,400 lb. sulphuric acid @ \$0.029, . . . . .	\$69.60
363 lb. borax-glass @ \$0.235, . . . . .	85.30
139 lb. soda @ \$0.03, . . . . .	4.17
75 yd. filter cloth @ \$0.21, . . . . .	15.75
1 ton coal @ \$3.50, . . . . .	3.50
1 ton coke @ \$9, . . . . .	9.00
6 No. 100 graphite crucibles @ \$6.56, . . . . .	39.36
Miscellaneous, . . . . .	5.00
Total supplies, . . . . .	231.68
Total cost of melting, . . . . .	\$305.18
<i>Leading Slag and Refining Matte.</i>	
Labor, 4 days @ \$4, . . . . .	\$16.00
1 day @ \$3, . . . . .	3.00
Total labor, . . . . .	\$19.00
Supplies, 227 lb. soda @ \$0.03, . . . . .	6.81
100 lb. litharge @ \$0.08, . . . . .	8.00
½ ton coal @ \$3.50, . . . . .	1.75
½ ton coke @ \$9, . . . . .	4.50
Power for air, . . . . .	2.00
Total supplies, . . . . .	23.06
Total cost of leading slag and refining matte, . . . . .	42.06
Total cost, . . . . .	\$347.24
Total oz. bullion, . . . . .	1567.6
Cost per oz. of bullion, . . . . .	\$0.221

\* NOTE.—Since writing the above, the costs have been very much reduced. A recent clean-up showed a total cost of \$0.129 per ounce of bullion produced.—G. H. C.

Mr. Tavener does not think a preliminary acid-treatment necessary; but Mr. D. J. Williams,<sup>18A</sup> in the discussion of Mr.

<sup>18A</sup> Discussion of the Lead-Smelting of Zinc-Gold Slimes, *Journal of the Chemical and Metallurgical Society of South Africa*, vol. iii., No. 7, November, 1902, p. 103.

Tavener's paper, argues that the precipitates should receive acid-treatment before being smelted with lead. He gives the following objections to smelting when zinc is present:

"(a) A great deal of experience is required to handle zinc in a furnace. . . .

"(b) . . . this process is not one of oxidation, but the zinc must be fluxed. If oxidation sets in then, I maintain there will be a loss of gold and silver.

"(c) Before all the zinc is fluxed, I should imagine that some would float on the surface of the molten mass, and carry by far the greater part of the gold and silver, as in Parkes' process of de-silverizing and de-aurizing lead bullion. If the zinc is allowed to oxidize or burn, undoubtedly there will be a great loss by volatilization. . . ."

Various other arguments have been urged for acid-treatment prior to lead-smelting.

#### IV. THE REMOVAL OF ZINC BY DISTILLATION.

It seems desirable to remove zinc more completely than can be done by acid-treatment, and also to obtain it in a form at once available for precipitation again. I feel safe in saying that from 40 to 50 per cent. of the zinc originally used for precipitation could be thus recovered.

Philip Argall<sup>19</sup> says, in speaking of the practice at Cyanide, Colo., upon the Cripple Creek ores:

"Our general practice shows a consumption of 0.92 lb. of zinc for each ounce of fine bullion produced. Of this amount not more than 40 per cent. is dissolved, [by the cyanide solution], the remaining 60 per cent. being removed from the [precipitation]-boxes with the precipitate at each weekly clean-up, and owing to its fine state of division and richness in gold we find it preferable to destroy the zinc and recover the gold." Note.—The bracketted words are mine.—G. H. C.

In some cases the amount of fine zinc removed with the precipitates is less than that stated by Mr. Argall; in many others, more.

It has seemed to me that this material could be retorted, like the zinc crusts from the Parkes process; a certain percentage of lead and charcoal being mixed with it—the lead, to form an alloy with the gold and silver present, and prevent loss; the charcoal, to reduce zinc oxide to metallic zinc, and also carbon dioxide to monoxide, thus keeping the interior of the retort and condenser reducing. At a certain temperature below the melting-point of zinc, the zinc vapor would condense

<sup>19</sup> *Mineral Industry*, vol. vi., pp. 378-379.

as dust; and if the zinc vapor be diluted with some inert gas, the formation of dust is greatly aided. Air would have to be completely excluded from the condenser, since the zinc dust, when hot, is very easily oxidized; and, for use in precipitation, it should contain as little as possible of the oxide, which not only does not precipitate gold and silver but actually consumes cyanide. With proper precaution, practically all the zinc vapor driven off could be recovered as zinc dust, at once available as a precipitant.

When ores containing mercury or tailings from amalgamation-processes are treated by the cyanide process, the precipitates contain mercury. In several cases the mercury contained in the precipitates has been profitably recovered, by retorting the precipitates in an ordinary mill-retort. If zinc were removed by distillation any mercury present could be condensed at the outset with little extra trouble; and in this way mercury could be recovered from precipitates which could not be economically retorted for mercury alone.

Sulman and Teed<sup>20</sup> have retorted the precipitates obtained by their bromo-cyanide process. The practice at Deloro, Can., however, as described by Mr. Bosqui, produced very base bullion, containing about 69 per cent. of gold, and from 25 to 29 per cent. of zinc.

An attempt made at the Golden Gate mill, Mercur, Utah,<sup>21</sup> to distil off the zinc in Faber du Faur furnaces, was unsuccessful because of the impossibility of reducing the large amount of zinc oxide present, and "the persistent retention" in the previously muffle-dried charge, of "combined water, which caused explosions in the furnace." This test seems far from conclusive.

The trouble seems to have been insufficient heat. The explosions attributed to combined water were probably due to the formation of a crust on top of the charge, due to lack of heat. It has been found in distilling the zinc crusts from the Parkes process that it is necessary to maintain the temperature well above the boiling-point of zinc until distillation is complete; otherwise crusts form and explosions occur.<sup>22</sup>

<sup>20</sup> *Journal of the Society of Chemical Industry*, December 31, 1897, pp. 961 to 970.

<sup>21</sup> *Mining and Scientific Press*, May 5, 1900, p. 492.

<sup>22</sup> *Metallurgy of Lead*, Hofman, 6th ed., p. 490.



*Experimental Distillations.*

Twelve hundred grams of cyanide precipitates were taken for this experimental work.

A complete analysis of the precipitates gave the results shown in Table IV.

TABLE IV.—*Analysis of Precipitates Used for Distillation-Experiments.*

Element.	Per Cent.	Remarks.
Gold.....	16.639	Determined by $\text{Fe}_2(\text{SO}_4)_3$ method.
Silver.....	23.166	
Iron.....	0.170	
Aluminum.....	0.800	
Calcium.....	6.680	
Magnesium.....	0.070	
Zinc (as metal).....	14.510	
Zinc (as oxide).....	18.060	
Lead.....	0.730	
Arsenic.....	.....	
Antimony.....	.....	
Mercury.....	.....	
Cadmium.....	trace	
Copper.....	0.870	
Manganese.....	0.100	
Silica.....	1.760	
Sulphur.....	0.210	
Carbon dioxide.....	3.770	
Moisture.....	.....	
By difference,—cyanide, oxygen, alkalies and matter not deter- mined.....	12.465	
Total.....	100.000	

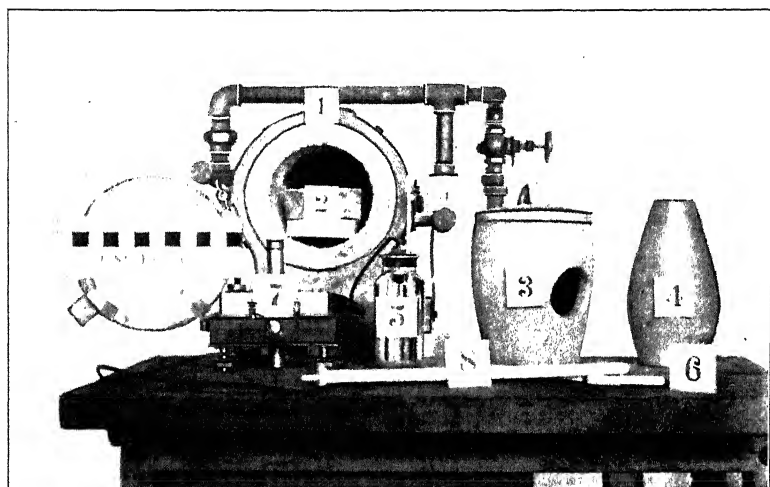
The distillations were made in a small graphite retort,<sup>23</sup> the counterpart, on a small scale, of those used in practice for re-torting the zinc crusts from the Parkes process. The retort was heated in the cylindrical gas-furnace belonging to the equipment of the metallurgical laboratory of Columbia University.

The retort was connected by means of the section of a small graphite crucible (which acted as an allonge), with the condenser, which was a large graphite crucible. Figs. 1 and 2 show the whole apparatus and a section of the furnace.

The purpose of the experiments was to determine:—

<sup>23</sup> Made to order by the Jos. Dixon Crucible Co., and furnished gratuitously by Mr. J. A. Walker, Vice-President of the company.

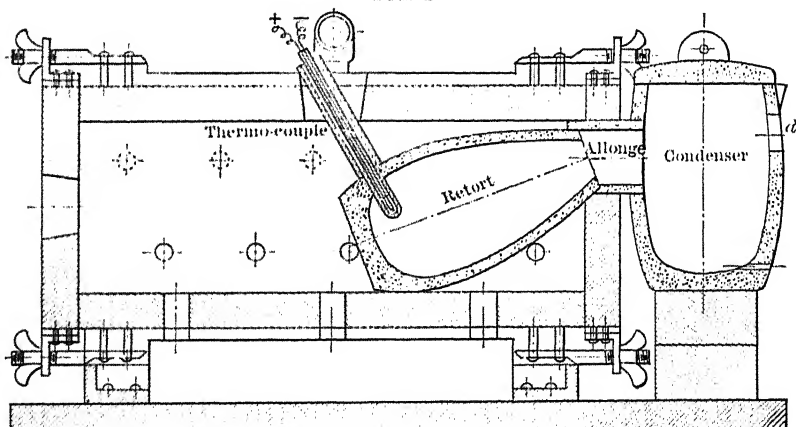
FIG. 1.



- |   |   |
|---|---|
| 1. Furnace used for heating retort and cupel. | 6. Junction of thermo-couple.   |
| 2. Test or cupel.                             | 7. Siemens galvanometer.  |
| 3. Graphite crucible, used for condenser.     | 8. Porcelain tube for protecting couple from zinc vapor.                          |
| 4. Graphite retort.                           | 9. Union, where gas was shut off from lower set of burners while using the cupel. |
| 5. Cold junction of thermo-couple.            |   |

APPARATUS USED FOR DISTILLATION EXPERIMENTS.

FIG. 2.



LONGITUDINAL VERTICAL SECTION OF DISTILLATION FURNACE, RETORT AND CONDENSER.

1. Whether the zinc could be removed by distillation.
2. Whether the losses of gold and silver by volatilization would be prohibitive.
3. Whether the zinc could be recovered in a form available for use as a precipitant.
4. What temperature was necessary to remove the zinc completely.
5. What time was necessary to remove the zinc; and also minor factors of the problem.

*Experiment I.*—A mixture of 300 g. of precipitates with 500 g. of test-lead and 50 g. of charcoal was placed in a small graphite retort, and this in the gas-furnace. The gas was then turned on, and the retort allowed to heat up slowly for one hour until all moisture was driven out, then the heat was raised and the heating continued for five hours.

The weight of zinc recovered from the condenser was 74 g., or 85 per cent. of the total zinc present. The zinc contained by assay 7.2 oz. of silver and 0.8 oz. of gold, per ton. This experiment seemed to establish the fact that the zinc could be driven off satisfactorily by distillation, in fact more completely than by acid-treatment. Also that losses by volatilization were not high, and in view of the fact that the zinc was to be used again for precipitation, they were of no moment.

*Experiment II.*—A mixture of 400 g. of precipitates with 100 g. of charcoal and 1,200 g. of granulated lead was treated as in the first experiment; except that a thermo-couple was introduced into the retort as shown in Fig. 2, and temperatures were taken every 15 minutes, as shown in Table V.

This experiment showed a smaller loss by volatilization than No. 1, due probably to the greater percentage of lead in the charge. The assay of the metallic zinc was rather high; but this is due to the fact that it collected in the allonge, and was probably enriched by particles of precipitates which lodged there while charging the retort.

*Experiment III.*—This was undertaken to ascertain whether zinc could not be completely driven over at a lower temperature than that used in I. and II., and what function time and temperature played in the expulsion of the zinc.

A 200-g. sample of the precipitates, mixed with 200 g. of test-lead and 50 g. of charcoal, was retorted, the temperature

TABLE V.—*Results of Experiment II.*

Time.	Temperature. Degrees C.	Remarks.
10:30	22	Time of starting.
10:45	159	
11:00	450	First signs of zinc.
11:15	671	
11:30	874	
11:45	1011	
12:00	1100	
12:15	1125	
12:30	1206	
12:45	1269	
1:00	1287	
1:15	1310	
1:15	1310	
1:45	1362	
2:00	1382	
2:15	1400	
2:30	1414	
2:45	1420	
3:00	1382	
3:15	1400	
4:00	1414	No temperature taken at the last two points.
4:15	.....	
4:30	.....	

being taken as before at intervals of fifteen minutes. A sample of the material in the retort was removed every half-hour and assayed for zinc. The results of these tests are given in Table VI.

NOTE.—Much trouble was experienced in introducing the thermo-couple into the retort in such a way that it is not injured by the zinc vapor. In the first experiment the couple was introduced as shown in Fig. 2, but this was found unsatisfactory, as upon strongly heating the retort the short clay tube became loose in the retort and zinc vapors escaped. The plan finally adopted, which has given no trouble, is to enclose the wires of the couple in a porcelain tube having a high fusing-point, and long enough to reach outside of the furnace. The closed end is introduced into the retort about two inches. The space between the porcelain tube and the walls of the retort is packed with asbestos, which is found to do the tube less harm than any other substance which would withstand the high temperature. This joint of course is not impervious to zinc vapor, but as the collecting of the zinc in this experiment was of secondary importance that disadvantage was of no moment.

*Experiment IV.*—A 150-g. sample, mixed with 40 g. of charcoal and no lead, was retorted in this experiment; the temperature and a sample of zinc being taken every fifteen minutes. Table VII. shows the results.

In this case no assays were made of the condensed zinc,

TABLE VI.—*Results of Experiment III.*

Time.	Temp. Degrees C.	Per Cent. of Zinc in Charge.	Per Cent. of Zinc Expelled.	Remarks.
11:15	20	11.58	0.0	First zinc appeared at 671.
11:30	498	.....	.....	
11:45	866	10.51	9.2	
12:00	1055	.....	.....	
12:15	1055	9.72	16.1	
12:30	1119	.....	.....	
12:45	1166	6.95	40.0	
1:00	1185	.....	.....	
1:15	1197	4.41	61.9	
1:30	1204	.....	.....	
1:45	1206	1.48	87.2	
2:00	1210	.....	.....	
2:15	1210	1.82	84.2	
2:30	1250	.....	.....	
2:45	1225	0.53	95.4	
3:00	1265	.....	.....	
3:15	1275	0.38	96.7	
3:30	1284	.....	.....	
3:45	1284	0.19	98.3	
4:00	1281	.....	.....	
4:15	1281	0.10	99.1	

TABLE VII.—*Results of Experiment IV.*

Time.	Temp. Degrees C.	Per Cent. of Zinc in Charge.	Per Cent. of Zinc Expelled.	Remarks.
1:00	22	24.50	0.0	First zinc appeared at 750.
1:15	820	24.28	1.0	
1:30	1015	23.90	2.4	
1:45	1134	28.67	3.3	
2:00	1168	17.77	27.4	
2:15	1194	5.41	77.9	
2:30	1194	4.98	79.6	
2:45	1209	4.50	81.6	
3:00	1214	2.40	90.2	
3:15	1209	1.72	92.9	
3:30	1204	1.92	92.2	
3:45	1221	2.40	90.2	
4:00	1229	1.00	95.9	
4:15	1239	0.68	97.2	
4:30	1264	0.68	97.2	
4:45	1274	0.62	97.4	
5:00	1264	0.43	98.2	
5:15	1290	0.38	98.4	
5:30	1274	0.38	98.4	
5:45	1280	0.19	99.2	
6:00	1296	0.10	99.6	

as they would be of no value; for even when the utmost care was exercised in removing the samples for the zinc-determination, small particles would fall off into the condenser, thus

causing the condensed zinc to show much higher values in gold and silver than could occur by volatilization.

*Experiment V.*—A mixture of 95 g. of precipitates with 95 g. of borax-glass, 400 g. of test-lead, 25 g. of soda, 10 g. of silica and 35 g. of charcoal, was retorted as in the previous experiments. The thermo-couple was not used in this experiment, as it was feared that the fluxes used would attack the porcelain tube which protected it.

The object of this experiment was to determine whether a fusible slag could not be formed which could be easily poured from the retort and from which the lead would separate. But although there was a large excess of fluxes used, the material came out as a sintered mass, probably by reason of the excess of charcoal in the charge.

The residues after distillation showed 0.14 per cent. of condensed zinc, 3.86 oz. of gold, and 35.8 oz. of silver, per ton.

The total time of the experiment was five hours, and about the same temperature was maintained as in Experiment IV.

#### *Discussion of the Results of the Distillation-Experiments.*

The retorting of the gold-silver-zinc precipitates, while resembling that of the silver-gold-zinc alloys obtained in the Parkes process, requires a greater proportion of charcoal by reason of the large percentage of zinc oxide, and also the carbon dioxide present. In both cases the best results are obtained by raising the heat to the proper degree at once and maintaining it at that point until distillation is completed.

The time required on a small scale has been 4 or 5 hours. About 90 per cent. of the zinc is removed in the first 2.5 hours and at a temperature not much exceeding 1,200° C. To remove the remaining zinc requires from 1.5 to 2.5 hours, and a somewhat higher temperature. It seems desirable to heat the material finally to about 1,300° C. in order to remove the last of the zinc.

The same thing is noted in retorting gold- and silver-amalgams. Most of the mercury is removed at a temperature not much above the boiling-point of mercury, but to remove the remainder requires a much higher temperature.

The volatilization of gold and silver is not excessive, and in view of the fact that the zinc can be used again for precipitation, it is of no importance. Indeed it is an advantage, be-

cause the zinc containing a small amount of gold and silver would be a very active precipitant.

I have not been able to determine exactly the conditions of condensation required to form a zinc dust suitable for use as a precipitant. It was noticed that the first zinc that came over, which was largely diluted with carbon monoxide, condensed as a dust fairly free from oxide. The condenser was cool at this time. A sample of this zinc dust showed 97.1 per cent. of metallic zinc. But later, as the zinc vapor became less diluted with carbon monoxide and the condenser became hotter, a portion of the zinc condensed as a liquid bath and the remainder as zinc dust, which in its highly heated condition rapidly oxidized to zinc oxide. There was always a chance for air to enter the condenser, as it was necessary to keep the orifice (*d*, Fig. 2) open a part of the time to permit the removal of accretions from the allonge. The result was that most of the product from these experiments was not fitted for precipitation.

The formation of the zinc dust seems to require that the zinc vapor be diluted with some inert gas (carbon monoxide is probably the best and most available) and condensed at a temperature below the melting-point of zinc, without access of air. This could be accomplished by condensing in a water-jacketed condenser, in which there was a stuffing-box through which a rod could be inserted, to keep the allonge free from accretions. Provision should also be made for introducing gas into the condenser, if it were found that the carbon monoxide formed in the retort was not sufficient.

The material in the retort at the end of the distillation is in appearance the same as at the start; but upon closer examination there will be found small shots of bullion all through it, and usually one or two large masses, especially if lead is used which very much increases the bulk of metal.

This mass can be poured from the retort like so much sand; no trouble is experienced from its sticking. The retort may be easily washed or cleaned by throwing in a little metallic lead while it is yet hot, and shaking the molten lead around. This seizes upon any particles of bullion sticking to the retort, and is subsequently poured and added to the main mass before it is run down.

The residues from the five experiments were united, mixed

with four or five parts of litharge, and melted down on a test. Enough iron filings and silica was added to make a fluid slag, which, after thorough fusion, was skimmed off, and the lead was cupelled. The amount of bullion recovered in this way was close to that shown to be present by assays on the original precipitates, although a little deficiency, due to the large number of samples taken for analysis, could not be avoided.

## V. GENERAL CONCLUSIONS.

Of the various methods of refining the gold-silver-zinc precipitates of the cyanide process, smelting with lead, after removal of the zinc, is the best.

Gold and silver are precipitated from cyanide solutions by zinc, as alloys or intimate mixtures of those metals with zinc.

It is possible to remove zinc more completely, and probably at a less cost, by distillation than by acid-treatment.

It seems possible to recover most of the zinc in a form at once available for use as a precipitant.

## VI. OUTLINE OF A PROPOSED METHOD OF TREATMENT.

Precipitates to be dried, mixed with lead and charcoal, and retorted. If mercury be present, it can be recovered at the outset of the process; later, zinc is condensed as dust, suitable for precipitation. The value of the zinc recovered would probably pay for the fuel used in retorting.

The residues containing the gold and silver to be smelted with lead in a furnace like the ordinary cupellation-furnace, iron-ore and silica being added in such proportions as to give a fluid slag, from which the lead separates. This is skimmed off and the lead cupelled in the usual way, the resulting litharge being used for the next operation.

The resulting bullion, after resmelting, to be cast into fine bars of gold and silver which need no further refining aside from parting.





# DISCUSSIONS.



## The Geological Features of the Gold Production of North America.

A Further Discussion of the Paper of Mr. Waldemar Lindgren (see *Trans.*, xxxiii., pp. 829, 1077).

H. W. TURNER, San Francisco, California (communication to the Secretary\*): Mr. Lindgren says that the gold-quartz veins of the Silver Peak region in Nevada occur chiefly in Paleozoic rocks, giving me as the source of information. It will appear from what follows that my statement to Mr. Lindgren must have been an incomplete one. While it is true that many of the gold-quartz veins near Silver Peak are in Paleozoic (chiefly Cambrian) rocks, yet the most important and productive veins are in pre-Cambrian gneisses and schists.<sup>1</sup> The only veins worked in 1900-1901 were those of Mineral Ridge, and these are chiefly in the Basement Complex that lies below the Lower Cambrian dolomites and limestones, and is presumably of Archean age. Some of these veins appear to be of pegmatitic origin, since they contain a little primary feldspar and muscovite; but the great vein of the Silver Peak Mining Company (the Drinkwater vein) and other producing-veins appear to be normal gold-quartz veins. The gneisses are sheared granites and quartz-monzonites, and the schists likewise are in part of granitic origin. The overlying dolomites, limestones and shales of the Lower Cambrian show little evidence of contact-metamorphism, or any sort of metamorphism. It seems likely, therefore, that the shearing that formed the gneisses took place before Cambrian time, and that the quartz-veins of Mineral Ridge may possibly be older than the Cambrian age.

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\* Received December 28, 1903.

<sup>1</sup> A description of the veins of the Silver Peak region is given in the *Pacific Coast Miner*, April 15, 1902.

## Flue-Dirt and Top-Pressure in Iron Blast-Furnaces: A Study of the Influences Controlling Them.

A Discussion of the Paper of Mr. F. Louis Grammer, p. 92.

FRANK FIRMSTONE, Easton, Pa. (communication to the Secretary\*): It may not be amiss to point out that the top-pressure in iron blast-furnaces is largely affected by variations in the temperature of the gas. I find in my note-books, cases in which the pressure was from 9.25 to 9.5 in. of water when the material in the furnace was low (after dinner-hour), which fell to from 8 to 8.5 in., when the furnace was again full, and the gas consequently cooled. The chief cause for this effect is, that, at the higher temperature, there is a decrease in the weight of the gas in the descending-leg of the siphon formed by the down-comer and the chimneys.

It does not seem probable that there is any direct relation between top-pressure and quantity of flue-dirt. Top-pressure, at any given moment, measures the "head" necessary to discharge the gas then passing, against the resistances of pipes, valves, etc., in the gas-conduits between the top of the furnace and the mouths of the chimneys, and can be increased at will by closing, more or less, the valves, or dampers, at the stoves and the boilers. The question of relation between top-pressure and flue-dirt can be settled very easily by direct experiment, working alternately with gas-valves open, and partly closed, and noting the top-pressures and the respective quantities of flue-dirt collected.

The quantities of fine ore blown away, as shown by Mr. Grammer's tables, are astonishing to one who has had no actual experience with Mesabi ore and hard driving. Flue-dirt, in the sense in which he uses the term, was unknown at anthracite furnaces so long as the working was normal; when it did occur, it was always an incident of scaffolding and bad work, and was no doubt caused by an increase in the velocity

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\* Received June 17, 1903.

of the gases, due to diminished cross-sections in the partly obstructed furnace.

F. LOUIS GRAMMER, Baltimore, Md. (communication to the Secretary\*): Perhaps I unduly emphasized the word "pressure." Of course it is the velocity and the volume of the gas and the fineness of the ore which affects, for the most part, the quantity of material blown out of the furnace. The top-pressures may be used as a practical guide, or partial index, of the increase in the velocity of the gases. If this pressure be increased by throttling the boiler-burners and the stove-valves, it would not be an index of increased velocity of exit through the down-take opening.

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### Water-Hoisting in the Pennsylvania Anthracite Region.

Discussion of the Paper of R. V. Norris, p. 106.

G. A. BURR, Parral, Chihuahua, Mexico (communication to the Secretary†): I regret that Mr. Norris did not give more attention to the hoisting of water in inclined shafts or slopes: the only slope mentioned being Hickory Ridge Slope, No. 4, of the Union Coal Co. The surface arrangement, illustrated in Fig. 12 of his paper, if not unpractical, is at least unscientific and cumbersome. It is my opinion that the back- and side-guides shown in Fig. 12 are not only unnecessary where rails are used as a track, but are unsafe, excepting in the case of a very firm hanging-wall, or roof, and sides of vein-matter, having absolutely no tendency, either to settle from the back, or to squeeze from the sides, an effect which would inevitably force the guides out of alignment.

The accompanying illustration, Fig. 1, shows what I consider a better method of hoisting water than that described by Mr. Norris. This method is based on the general practice in metalliferous mines of the West, where skips are used in slopes; and, while it is not original with me, I have frequently used it with very satisfactory results for dumping both ore and water.

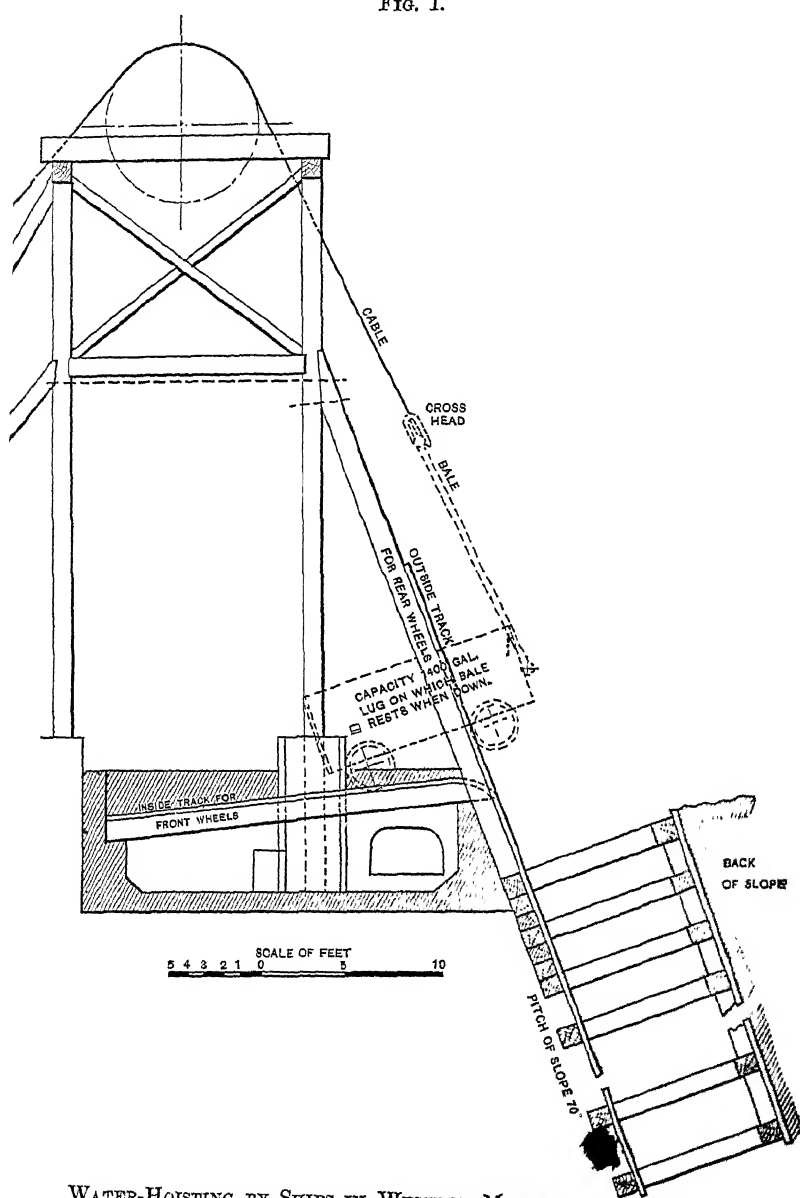
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\* Received July 13, 1903.

† Received April 13, 1903.

A special point to be observed is, that, as the slope approaches the vertical position, either the skip should be length-

FIG. 1.



#### WATER-HOISTING BY SKIPS IN WESTERN METALLIFEROUS MINES.

ened in order to have the center of gravity nearer the forward wheels, or the box should be depressed by raising the axles to

its sides (instead of placing them underneath), in order to have the center of gravity nearer the rails, and thus avoid the danger of having the front wheels lift from the rails, which would result in derailing the skip.

I have been greatly interested in Mr. Norris's paper on account of my own experience in hoisting water in metalliferous mines, where the usually limited shaft-area frequently precludes the hanging of a sufficient number of sinker-pumps to hoist the water.

The difficulty of solving this problem increases with the recession of the water-level from the place of water-discharge, and the consequent greater lift and increased duty required of the pumps. The use of tanks, or bailers, appears to me to afford a very practical solution of the problem, especially in cases where all of the pumps, including the "sinkers," have been "drowned." Here the tanks can be swung into the hoisting-compartments, the level of the water lowered, and the pumps recovered (provided, of course, that the tanks and the engines operating them are of sufficient capacity) without the inconvenience attending the tedious method of lowering steam- and water-columns, length by length, or of breaking joints and lowering the pumps.

For this method of hoisting water, where steam is used, it is needless to say that quick, direct-acting hoists, in competent hands, possess great advantages over those of the geared type.

C. F. HEATHCOTE, Beaconsfield, Tasmania (communication to the Secretary, New York meeting, October, 1908): Mr. Norris has made out a very strong case for bailing-tanks versus steam-pumps under certain definite conditions, but under those conditions only. The advantages of bailing from a drowned-out mine cannot be disputed, for when this occurs, there is no output, and the usual winding-shafts and engines are always available; but for dealing with large quantities of water when shaft-sinking, it is out of the question. The economies shown by Mr. Norris are due, less to the method of water-hoisting than to the substitution of one plant for six separate pumping-stations in use at the Short Mountain colliery, and to the water-hoists [being of ample power, while the pumps were often, apparently, fully loaded with no margin for overhaul and repairs.



The new pumping-plant, figured on presumably two stations, shows up well, while the Hazleton shaft with one (?) shows a much better result.

For further comparison, the result of six-months' work of a Duplex, 24-in. Plunger, Cornish plant, are given with those of Mr. Norris' in Table I.

TABLE I.—*Cost of Pumping 1,000 Cubic Feet of Water a Height of 1,000 Feet.*

Name.	Speed.	Cubic Feet of Water.	Height of Lift.	Cost.	
				Coal at \$0.50 Per Ton.	Coal at \$1.82 Per Ton.
Short Mountain colliery (old)....	Part speed.	208,191	1,141	\$0.3712	1.4757
Short Mountain colliery (old)....	Full speed.	409,823	1,093	0.2920	1.3239
Short Mountain colliery (new)....	Part speed.	.....	1,200*	0.2250	0.6866*
Short Mountain colliery (new)....	Full speed.	500,000*	1,200*	0.1650	0.6206*
Hazleton shaft.....	Full speed.	600,000	560	0.1674	0.6290
Fidler hoist.....	.....	337,200	960	0.2400	1.0049
William Penn hoist.....	.....	280,000	727.8	0.2180	0.9429
Lytle hoist.....	.....	500,000	740.6	0.2100	0.9749
Cornish plant.....	Part speed.	372,940	687	0.2187	0.8647
Cornish plant.....	Full speed.	571,390	687	0.1567	0.5026

\* Calculated.

This plant has been in use at the Tasmania Gold-Mine, Beaconsfield, Tasmania, for 13 years, the average height of lift to which the figures relate being 687 ft., with 5 per cent. valve-slip allowed. The total costs are made up as follows:—

	Part Speed.	Full Speed.
All wages, stores and repairs, . . . . .	\$0.1787	\$0.11663
Fuel, . . . . .	0.04003	0.04003

For anything below 1,000 ft., Cornish plants of this size become unwieldy and are out of the question, while it must be admitted by anyone who has had experience with the timber-trouble due to steam in mine-workings, that, with coal at \$0.50 per ton in unlimited quantities, water-hoist expenses would, in the end, work out the cheapest. This trouble, however, does not occur with electric power.

Table I. also shows the great bearing that local conditions have (especially the cost of fuel) on the relative advantages of different plants. The coal-consumption is not given for the Hazleton plant, and calculated plant, and in the above table the costs are from the six-months' run under adverse circumstances,

calculated, of a Riedler plant delivering an average of 11,469 cu. ft. of water a day, a height of 600 ft. The coal-consumption at \$4.82 per ton of 2,000 lb. having been \$0.51415 per 1,000 cu. ft. of water raised 1,000 ft.

R. V. NORRIS, Wilkes-Barre, Pa. (communication to the Secretary\*): In regard to Mr. Burr's discussion, the hoisting of water on slopes in skips or tanks is quite usual as an emergency expedient in the anthracite region, but, as far as I know, the Hickory Ridge plant is the only one designed for permanent use; the guides which Mr. Burr criticises are only at the bottom where the tank strikes the water, and the roof and sides are absolutely solid, as he suggests they should be.

There was no intention, as Mr. Heathcote seems to infer, of prescribing water-hoisting as a panacea for mine-water under any and all conditions, but an attempt was made to describe the practice as now in use, and to compare it with the ordinary pumping practice of the region.

Where fuel is very high, it unquestionably pays to consider it to the fullest extent, and under such conditions it might be advisable in water-hoisting to use multiple-expansion condensing-engines and, possibly where the water was good, a balance-rope, instead of the simple direct-acting engines carrying steam nearly full stroke, which were described.

The 56,000,000 foot-pounds duty per 1,000 lb. dry steam, to which Mr. Heathcote's Cornish plant seems to figure, appears rather high for a non-condensing plant, and leads me to assume that his pump is probably of the simple condensing type, while I should think that his high price of fuel would warrant a high-duty pump of, perhaps, 120,000,000 foot-pounds duty.

In my opinion, except for the advantages noted on page 127 of this volume, water-hoisting has no economical advantage over pumping at depths within the proper lift of one pump, its special sphere being in handling large volumes of water from considerable depths, especially where there is danger of flooding at certain seasons of the year.

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\* Received Jan. 11, 1904. See Preface to this volume.

### Electrical Apparatus for Coal-Mining.

A Discussion of the Paper of W. B. Clarke, p. 134.

W. L. SAUNDERS, New York City (communication to the Secretary\*) :—Notwithstanding the sweeping statements made by Mr. Clarke in this paper, the friends of compressed air are not dismayed. There is no war between electricity and compressed air. Each one has its field of usefulness, and in that field each is supreme. In spite of commercial interests, in spite of prejudices and the enthusiasm of the inventor, the thing that is best will survive and flourish.

At least ten years ago, our engineering papers contained large advertisements and pages of reading-matter about the application of electricity to mines, recording the introduction of numerous rock-drilling plants, and claiming that the problem had been solved, greatly to the advantage of the miner and with promise of large profits to the manufacturers of electrical apparatus. Edison and Thomson labored to apply electricity to mines, and each believed that the probabilities of success were great. This was the period of the solenoid rock-drill, which was admirably suited, theoretically, to the percussive principle and in the development of which hundreds of thousands of dollars have been spent without satisfactory result. I have searched in vain for a standard mining equipment where electric rock-drills are in constant, commercial operation. During a recent trip made from New York to California, I visited most of the important mines of the West and failed to discover a single electric equipment, so far as this class of mining-work is concerned. If any one will inform me where there exists to-day an electric-drilling equipment in constant service in mines in the United States, I shall be glad to receive the information.

Mr. Clarke has been for some years an active and intelligent writer on the subject of electricity for mining. Whenever his

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\* Received May 7, 1903.

papers have appeared, we have read them with interest; for in many cases his arguments in regard to the operation of electric motors have been used effectively as arguments in favor of compressed air, especially in connection with mechanical haulage. Mr. Clarke mentions the adaptability of electric apparatus to coal-mines, and the high efficiency of the electric system. There is no place where the pneumatic engineer has an easier time when competing for the introduction of machinery than in a coal-mine. Air is exactly what is wanted in such a mine, and electricity is what is not wanted. The one is safe and healthful, the other dangerous and destructive. The following extracts are taken from the Report of the Pennsylvania Department of Internal Affairs for 1899, Part V., of the Report of the Bureau of Mines:

"Besides the increase in danger from explosive gases, other elements of danger have been introduced into the mines by the use of mining-machines and electricity. These have been introduced during the past ten years, and it is the opinion of the writer that the use of electricity in any form in coal-mines is a menace to life, limb and property."

The report for 1901 says:

"Electricity is one cause of fatalities in the bituminous mines (seven having lost their lives through it during 1901) that so far has not proved fatal to any person in the anthracite mines. Electricity in various forms has been the cause of many deaths in the soft coal-mines, either from the men coming in contact with the electric trolley-wire or with the electric wire that carries the power to the electric cutting-machines. In my opinion, separate traveling-ways should be provided for the workmen when the haulage is done by electricity, unless the wires can be raised to a distance of at least six feet from the rail, and even then there should be sufficient room for passing on the main haulage roads at all points, as men cannot always reach the 'safety-holes' in time. In every case where electric-machines are used for cutting coal, the wires should be made absolutely safe, as men in the hurry of their work forget about the 'deadly wire,' touch it, and all is over, and the report follows, 'killed by an electric shock.' Humanity demands protection for the workingmen from this most deadly agent recently introduced and employed in coal-mines. I hope the time will come when 'compressed air,' 'liquid air,' or some other agent will supplant electricity in coal-mines.

"In gaseous mines, electric cutting-machines or other electric motors should never be permitted in use, as otherwise, sooner or later, they will be the cause of a great catastrophe."

It cannot but be of interest to the members of the Institute to read the following statement of "The Difficulties Which Must Necessarily be Met, and Some Means of Reducing the

Dangers of Faulty Insulation," which formed the subject of an admirable paper, in *Mines and Minerals* for November, 1900, by Mr. Alton D. Adams:

"In but few other places are so many conflicting requirements to be met, as to electric wiring, as in mines. Distances over which the electric energy must be distributed are often so great that comparatively high voltages are necessary to economy in the copper-conductors. The higher the voltage of circuits, the greater the necessity for a high degree of insulation, and yet the conditions in mines are such that most wires must be run bare. If wires of comparatively high voltage must be used without an insulating covering, it is desirable that they be erected beyond the easy reach of men and horses, but the head-room in mines is often such that this precaution cannot be taken. The large amount of moisture present in a mine renders it very difficult to maintain any high degree of insulation on wires unless they have a covering of rubber, and the sulphur present in the moisture of many mines, especially those of coal, rapidly ruins rubber insulation. The result is that circuits of comparatively low pressure are frequently run with bare wire. A moderate amount of insulation from the earth is obtained at the points of support by using only large porcelain or glass insulators for the attachment of the wire. Care should be taken to have the wires touch nothing save their insulating supports, and porcelain tubes are convenient in some cases for this purpose, where the sides or ceilings of passages are uneven. Such tubes should always be used in every case where wires go through partitions. With bare copper-circuits, in mines where there is much moisture, the most careful work will not insure a high resistance to earth, because of the films of moisture that collect on the surfaces of the porcelain and glass. While this moisture on insulators is not usually sufficient in amount to cause any serious loss of power, it may well cause a severe shock to one who makes a single contact with the circuit, if his body is also in electrical connection with the ground. The existence of these conditions points to the conclusion that all distribution circuits in mines should be operated at a voltage that is not ordinarily dangerous to the life of man or beast, so far as possible.

"Three kinds of electric service must usually be provided for in mines, lighting, stationary motive-power and hauling. The voltages of circuits for these purposes are apt to vary somewhat according to the kind of electrical supply adopted. The only satisfactory motors available for traction purposes are those of the direct-current type. Such motors may be operated at any voltage within wide limits, but the prevailing pressures for railways in mines are 250 and 500 volts. Stationary motors of the direct-current type are usually operated at 220 or 500 volts to correspond with generators for incandescent lighting or with the traction system. Satisfactory induction-motors, operated by two- or three-phase alternating-currents, are now in use. These motors are usually supplied by transformers, distant not more than a few hundred feet, and there is, therefore, no reason for a voltage higher than 250 on their local circuits. If incandescent lamps are supplied with direct current, the voltage on the two-wire system must be about 125 or 250 to correspond with those of lamps. In case the three-wire system of distribution is employed for the circuits, their maximum pressures will be twice those just named, or 250 and 500 volts, according to the pressure required at lamps. Lamps on alternating-current circuits are usually supplied by comparatively near transformers, and there is slight incentive to use maximum pressures greater than 125 volts on the two-wire, or 250 volts on the three-wire, system.

"With these facts, as to the general requirements and limits for electric pressures in mine-work, in mind, expedients for greater safety and convenience of operation may be considered. The standard pressure for electric street-railway lines all over the country is 500 to 600 volts. The Edison three-wire system of distribution, as extensively used by central stations, has very generally employed a pressure of about 250 volts between its outside wires. Experience, gained from these two extensive classes of circuits, has demonstrated that 500 volts is sufficient pressure to readily kill a horse and even a man, when a good contact with both sides of a circuit is made. On the other hand, it has been equally well demonstrated that 250 volts will not kill a man under any conditions as to contact that are at all likely to exist in practice, but this conclusion does not follow as to a horse. In mines, where both men and mules are liable to make contacts with live circuits of bare wire, it seems highly desirable that their voltages be limited, if possible, to 250, and especially that the difference of pressure between either side of any circuit and the ground be not greater than this amount. As above pointed out, there is seldom any good reason for a pressure of more than 250 volts on the alternating-current circuits for incandescent lamps and motors in mines. Direct-current distribution to both incandescent lamps and stationary motors may be carried on in mines at maximum pressures of either 250 or 500 volts, with corresponding possible differences of pressure between any wire and the earth of 125 or 250 volts respectively. This result is reached by the expedient, very generally adopted in Edison city systems, of grounding the third or so-called neutral wire of the three-wire system. Accidental contact is seldom made with more than one electric wire at the same time, and a shock, from one side of a three-wire system with a grounded neutral, is given with only one-half of the maximum voltage. The conditions as to the insulation of circuits in mines, as pointed out, are such that the maker of contact, with one side of an electric circuit, is quite apt to receive the maximum pressure of the circuit unless it has a grounded neutral. The moderate pressures necessary on transformer circuits and the grounded neutral of three-wire systems thus greatly reduce the dangers from circuits for lighting and stationary motors in mines.

"Most dangerous among electric systems in mines are the trolley or railway circuits. The wires leading to lamps and stationary motors may be grounded, so that a person touching either wire will receive the maximum pressure, though this is not usually as much as 500 volts. There is, however, some chance of escape without injury from accidental contact, since the ground connections on the lamp or stationary motor-circuit may be slight, or the electric pressure of the circuit may be low. One side of the street-railway system of electric conductors is, however, permanently and securely grounded by the use of the track as all or a part of the return-circuit. Any one touching the trolley- or the uninsulated feed-wires is, therefore, subject to the full difference of pressure between the two sides of the system, if they are in electric connection with the ground or rails. In places where the pressure on the electric traction system in mines is kept at about 220 volts, the probable damage to men or animals is very slight, but repeated and serious injuries have occurred in mines where 500 volts are employed. The very material incentive to the use of the 500-volt traction system is the fact that, compared with a system having 220 volts as a maximum, it saves 75 per cent. of the necessary copper in conductors, all other factors remaining constant. This consideration is of especial weight in those instances where distances are so great that the relative amount of the investment for copper is in any case large. Fortunately, however, there is a means, though it has not been generally employed for the purpose, by which the advantages of the 500-volt traction system

in mines can be retained, and at the same time the pressure between the trolley or any of its feeders and the earth or rails limited to 250 volts. This result is reached by applying the three-wire system, long successfully used for lamp and stationary motor-circuits, to traction-circuits in mines. For this purpose two trolley-wires should be used, one connected to each side of the traction generator, or to the free sides of two generators in series. If only one generator is used, the conductors from the rails to the power-station should be connected to a small dynamotor, designed to keep the rail-pressure half-way between that of the main terminals. If two generators are used in series for the traction lines, then the rail-conductors should be taken to their common connection. On a double-track road, one trolley-wire, representing one side of the system, should be erected over the center of each track, but for a single-track road one trolley may be located about over each rail. In this latter case, trolley-poles should be mounted near one side, instead of on the center, of cars or locomotives, and about one-half of them take current from each trolley. This arrangement evidently makes the rails the third or neutral wire of a three-wire system, and any accidental contact between either trolley-wire and the rail or ground is exposed to a pressure of 250 volts, when the voltage between the two trolley-wires is 500. Contact through accident with both trolley-wires at the same instant is improbable, and the three-wire thus offers a decided increase of safety over the two-wire system, with 500 volts between trolley and rails. While the low-pressure circuits for lamps and stationary motors are quite safe as to their own voltages, there is danger from another source in those cases where these circuits run from alternating-current transformers that are supplied by high-pressure lines. This danger arises from the fact that a ground connection at any point on the high-pressure circuit and a cross between the high- and low-pressure circuits inside the transformer or elsewhere, exposes one who makes contact from the low-pressure to the earth at any point to the voltage of the high-pressure circuit. Such an accident would probably result fatally, as the primary alternating lines usually operate at pressures as great as 2000 or 3000 volts. This danger can be very largely averted by a substantial and permanent connection between one wire of every secondary circuit and the ground.

"If the secondary circuits from transformer are on the two-wire plan, either side may be connected to earth, but if they are run three-wire, the neutral should be so connected. The connection to the ground when properly made is of much lower electrical resistance than the body of a man or animal, and the only damage likely to result from a cross between the primary and secondary circuits is a blowing of fuses. The high-pressure primary lines to transformers should be kept out of the parts of mines in which work is going on, as far as possible. Rubber-covered wires, cotton braided and with a sheath of lead on the outside, seem to be the best material for the primary circuits. Where high-pressure wires are exposed to mechanical injury, as in vertical shafts, the rubber- and lead-covering just mentioned should be used and a substantial iron-pipe erected, into which the leaded wire is subsequently drawn. It is not necessary to have any lining of insulation for the iron-pipes, or to supply insulating supports for it."

These are but a few of the many statements made from time to time by men of practical experience in coal-mines. Electricity, with all its usefulness and value to mankind, is not the best power to be taken down into a mine where gases and dust are liable to accumulate, and where human life is endangered

by the electric spark. In cases of this kind the question of efficiency is of little or no importance when compared with greater and more serious questions; and, in face of the facts with which the members of this Institute are familiar, I deny Mr. Clarke's statement that the coal-mine is "a field to which the electric motor seems to be especially adapted."

As to the pneumatic mine-locomotive, the statement is made that it "is not an unqualified success," and among the reasons stated is that "its radius of operation is restricted." A mine-locomotive which is absolutely restricted to the limits of a trolley-wire, would seem to have a more limited radius than one which is independent of any wire or connection, which carries its own power stored in tanks and which might go anywhere and on any track to the extent of its storage-capacity. Such is the pneumatic locomotive, the first of which for mining purposes was built in the late eighties and early nineties, and of which, probably, not more than six or seven were in use in 1895, while to-day there are at least 150 of these locomotives in successful operation in mines, 125 of them having been built during the last three years. This record will, I think, compare favorably with that which can be shown by the manufacturers of electric mine-locomotives.

For a more detailed study of this subject, I beg to refer to a very practical paper by Mr. J. H. Bowden, entitled "Notes on the Compressed-Air Haulage-Plant at No. 6 Colliery of the Susquehanna Coal Co.," and presented at the Canada meeting of the Institute in August, 1900.\* In this paper Mr. Bowden shows that, by the use of air-haulage as against mule-haulage, the total saving in two years was almost equal to the total cost of the air-haulage plant; and he goes on to say that "at the average rate of saving for 1897 and 1898, the entire cost of the plant would be saved in 361 working-days."

I am indebted to Mr. E. P. Lord, General Manager and Superintendent of the H. K. Porter Co., of Pittsburg, Pa., for the comparison of cost of haulage in coal-mines by means of compressed air and electricity respectively, given in Table I.

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\* *Trans.*, xx., 566.



TABLE I.—*Comparative Cost of Compressed-Air and Electric Haulage.*

	COMPRESSED AIR.	ELECTRICITY.	
	Actual.	Estimated.	Actual.
Number of working-days during year....	160	200	141½
Output per day, tons.....	(a) 2362½	989	989
COST PER DAY.			
Engineer, power-house.....	\$1.16	\$1.20	\$2.84
Motorman.....	4.20	4.23	9.31
Helpers (brakemen).....	3.20	3.20	3.61
Electrician.....	.....	1.67	3.68
Repairs to motors.....	0.74	5.95	8.42
Repairs to line.....	.....	.....	0.46
Repairs to generator.....	0.57	.....	0.61
Fireman.....	.....	.....	2.50
Depreciation.....	(b) 4.74	(c) 5.20	(c) 8.17
Interest.....	4.73	.....	4.41
Interest, repairs and depreciation, 174 H.P. boiler.....	1.63	.....	.....
Oil and waste for motor.....	0.25	0.22	0.35
Oil and waste for generator.....	0.47	.....	0.74
Steam (fuel and firing).....	2.32	.....	.....
Totals.....	\$24.01	\$21.67	\$45.10
Cost per ton.....	0.01015	0.02192	0.0456

(a) Tons of coal hauled.

(b) At 5 per cent.

(c) At 3 per cent.

NOTE.—The figures for compressed air represent the operation, in 1898, of the Glen Lyon plant, as described in Mr. Bowden's paper, above cited, and were taken from that paper, except as to the cost per ton, which has been calculated by the writer. Mr. Bowden's final estimate of cost is \$0.0193 per ton-mile. As he estimates 1245 ton-miles per day, or less than one mile of average haul, the cost per ton hauled is less than the cost per ton-mile. The ton-mile basis is not a fair one for comparison, because the delay at terminals forms so large a part of the time consumed by the locomotives. The time lost in this way would remain a fixed quantity, regardless of the length of haul; and, therefore, if the haul were longer the locomotives would make a better showing on the ton-mile basis.

The compressed-air locomotives under consideration were, respectively, the fourth and fifth ever built by the H. K. Porter Co., and were crude in many ways, as compared with the compressed-air locomotives now being built. We have no doubt that the electric locomotives built for the Hillside Coal & Iron Company were also inferior in many ways, as compared with the present product of the General Electric Co. But why should one type improve more than the other?

In this table, interest is calculated by Mr. Bowden at 5 per cent. for the compressed-air plant, and at 3 per cent. in the catalogue of the General Electric Co.

The figures for electricity were taken from Catalogue 1030 of the General Electric Co., entitled, "Electric Mine-Locomotives," and published August 6, 1901. The column headed "Actual" gives the results accomplished by the electric-haulage plant at No. 2 shaft of the Hillside Coal & Iron Co., Mr. W. A. May, Superintendent. Under "Estimated," these results are given as re-calculated upon the assumption of 200, instead of 141.25, days of operation during the year.

Conditions unknown to the writer might have a tendency to compensate partially for the great difference in cost per ton shown above; but it would be hard to find conditions sufficiently unfavorable to electric-haulage to remove that difference entirely.

As to efficiency, it is not denied that the efficiency of an air-locomotive, like the efficiency of all locomotives, is low. Probably, 20 to 30 per cent. is correct when applied to air-locomotives which do not reheat the air before use. It must be denied, however, that the efficiency of an electric-locomotive is 55 per cent. In figuring on haulage-plants for mines we have been told by reputable men that the estimates of boiler-capacity for the pneumatic plant were substantially the same as those furnished by bidders of electrical apparatus. In a paper written by Mr. Clarke for *Mines and Minerals*, in July, 1901, entitled "Electricity vs. Compressed Air—A Comparison of the Efficiencies and Relative Cost of Installation of the Two Systems," he makes the statement that 150 h.p. of boilers will be required to operate his supposed electrical installation, and that in the case of compressed air the steam-engine driving the compressor was actually indicating 150 h.p. Mr. Bowden makes the statement in connection with the paper which Mr. Clarke is criticising, that he estimates the steam-consumption at 174 h.p.; and, further, that a second compressor is being installed, from which Mr. Clarke infers that the final steam-consumption will be 300 h.p. This cannot be the case, as the present compressor is doing the work, and the installation of the second compressor will simply allow the original machine to be operated at below its normal capacity without any increase in the steam-consumption. But in no case would one be led to believe that there was a difference in efficiency of 20 per cent. for compressed air and 55 per cent. for electricity.

As to the relative cost of air- and electric-plants, Mr. Clarke says that a compressed-air plant "costs from two to three times as much as an equivalent electric mine-haulage system." With all due respect for Mr. Clarke, I must maintain that this state-

ment is incorrect. The relative cost of any two systems would vary with conditions. The facts are that, in putting in bids for pneumatic installations, we have found that at times the difference in cost has been from 50 per cent. in favor of compressed air to 50 per cent. against compressed air, according to the conditions which existed; and we may say in a general way that the cost of an electric installation is lowest when operating on a single entry, involving the minimum amount of wiring, whereas, when a number of diverging roads occur, the independent nature of the compressed-air locomotive, which allows it to operate over additional lines without any previous preparation or expense, places it in the most advantageous position.

In the early part of Mr. Clarke's paper he alludes to "weaknesses" which have developed in the use of electricity in mines, and goes on to say that this matter has received attention, and improvements have been made which have facilitated "the repairs and renewals." Later on in his paper, when comparing the pneumatic with the electric-locomotives, he speaks of the former as "relatively complicated" and "subject to very frequent repairs." This is rather rough on my friends in the air-line, especially since it is a fact that an air-locomotive is a simple machine, differing little from the well-known perfected mechanism of the common locomotive, and that it is very much more simple in its construction and very much less liable to repairs than an electric-locomotive. We have no better authority on this point than Mr. Clarke himself, who, in his paper in *Mines and Minerals* of July, 1901, previously cited, says that "the operating expenses of the electric-haulage system would not be much less than those of the compressed-air system."

It may be added that the H. K. Porter Co., in numbers of cases, although local conditions have made the initial cost relatively high, have received orders solely because an unprejudiced investigation made by disinterested parties has shown that the operating expenses of the air-plant, particularly for repairs, have been so much less than those of the electric-haulage system that the difference would pay a handsome return upon the initial investment.

Furthermore, I desire to call attention to a discussion in *Mines and Minerals* of Mr. Clarke's paper of July, 1901, in which the following correspondence occurs:

"(In order that the above discussion may be fully understood, and since Mr. Bowden, who prepared the paper which is used as the basis of Mr. Clarke's figures, has died since his paper was printed, we have submitted Mr. Clarke's figures to Mr. Bowden's successor as chief engineer of the coal companies of the Pennsylvania Railroad Co., Mr. R. Van A. Norris, whose answer is appended :)

"Mr. H. H. STOEK, Editor *Mines and Minerals*, Scranton, Pa. :

"Dear Sir: I return to you herewith the article, 'Mine-Locomotive, Electricity vs. Compressed Air,' by W. B. Clarke. I have made calculations similar to those made by Mr. Clarke, and believe his conclusions to be practically correct as far as they go. But there are other conditions besides economy which have decided us in favor of compressed-air haulage, the principal ones being the gaseous character of the mines, in which the possible sparking of an electric machine would be a source of grave danger; the simple character of the mechanism of a compressed-air locomotive, which is within the capabilities of a colliery plant to keep in order and repair; the advantage of having considerable radius of action beyond the charging-stations, and the beneficial effect of the air on the ventilation of the mine.

"As was shown in Mr. Bowden's paper, the cost of power is but a small fraction of the total cost of operation, the saving for even a 50-per-cent. decrease in the steam used being but .09 of a cent per gross-ton mile.

"As an evidence of our entire satisfaction with this method of haulage, I might add that we have just ordered another compressed-air haulage-plant complete for our No. 1 shaft, Nanticoke, Pa.

"Yours very truly,

"R. V. NORRIS,

"Chief Engineer."

The foregoing statement, made by Mr. Norris, refutes the argument that the pneumatic-locomotive is complicated. Mr. Norris's experience and knowledge of this subject are beyond dispute.

In connection with the matter of repairs, it may be well again to quote Mr. Clarke, who wrote as follows in *Mines and Minerals* for April, 1901 :

"*Electric-Locomotives—Things to be Observed in Choosing, Operating and Caring for Mine-Locomotives to Secure Greatest Economy.*

"Annoying delays are often experienced at mine-haulage plants, where there are several locomotives, due to frequent 'blowing' of the circuit-breakers. It is proper, of course, that the circuits should be automatically opened, when the generator is called upon to deliver a current beyond its safe capacity, but it is annoying if the service is interrupted with too great frequency. The remedy for this trouble also lies, to a very great extent, in the hands of the motorman, as the intelligent use of the controller will minimize the demand upon the generator. This trouble is aggravated by the simultaneous starting of all of the locomotives when the circuit-breaker is reset by the engineer at the power-house, since each locomotive requires a comparatively large current when starting, and the sum of their starting-currents is often sufficient to immediately blow the circuit-breaker again. In this way it often happens that the circuit-breaker is thrown out several times in rapid succession, and the entire haulage system may be interrupted for a

half hour or more. The engineer occasionally becomes righteously indignant, allows the circuit-breaker to remain out a few moments before resetting it; and the relations between the employes of the power-house and the motormen become strained, with no good results to the company."

In this paper, also, Mr. Clarke advocates the establishment, where a number of electrical locomotives are employed, of an inspecting force, which shall work all night in cleaning, hunting for defects and making repairs; and, further, that a spare armature or two should always be carried in stock, together with other material for repairs; and that this is necessary in order to avoid the delay and expense due to the loss of the use of the locomotive, the consequent reduction of the output, the express- and telegram-charges, not to mention the annoyance, that can be avoided if the locomotive is regularly inspected, and if extra parts are carried in stock at the mine. His summary at the end of this article (the seventh clause) reads:

"Last and most important, establish a rigid system of inspection and carry a *liberal stock of repair-parts.*" (The italics are mine.)

We might add to this a little advice to anyone who contemplates an electric installation: that he be prepared to take care of all repairs promptly; that he employ an experienced electrician; and that he have a shop sufficiently equipped to provide for such repairs as will be inevitable. It is well known that, in connection with the mines of the Pittsburg Coal Co., there are several shops, situated at central points, in which skilled mechanics are constantly making repairs to the electrical mining-equipment, and in the large number of mines controlled by the Pittsburg Coal Co., comprising 70 or 80 different mines, only two are equipped with air-haulage plants, one of the air-haulage plants having been installed thirteen years ago; yet it is stated by experts familiar with the operation of these mines that, in the matter of cost of repairs and safe uniform service, these air-plants have given less trouble and have cost less than any of the electric-plants at other mines.

There are, of course, places where electricity is particularly well adapted for mine-haulage, and where it is to be preferred possibly to any other means; but those places do not exist everywhere; and it is unfair to state the broad principle, that

electricity is better suited for mine-haulage than compressed air.

It is a common thing to hear from electrical sources that compressed air gives a low efficiency. A restricted definition of the word efficiency is, the proportion of the power, generated by the coal consumed under the boiler, which is delivered by the machine in question. But in the broad, practical sense, namely, that of capacity to accomplish work in the best manner consistent with all conditions, the advocates of compressed air claim that, as compared with all other powers, it is supreme in mines. It is common to criticise the efficiency of a percussive rock-drill. The catalogues of electric-drill manufacturers have invariably stated comparisons showing that an electric-drill consumes from 2 to 3 h.p. as compared with 8 to 10 h.p. in an air-drill. But, in the first place, the figures of electrical power-consumption are largely theoretical, while those of the air-drill are based on an experience of forty years. There is no doubt about the fact that an air-drill, when doing good service in a mine, will consume from 8 to 10 h.p., but in doing this it does its work; it gets the hole in; it makes progress; it costs the minimum amount in repairs, as compared with the number of linear feet of hole drilled. In these respects the air-drill is the most efficient machine at work in a mine to-day. Think for a moment of the hard work done by this "pounder." Here is a machine light enough for two men to handle, and yet containing within itself the capacity to make a hole in a piece of hard granite or trap-rock at the rate of from 2 to 5 in. a minute. The machine consists of little more than a piston, a valve and a cylinder, as these are the principal working parts. The power is led through a piece of hose (which, in itself, is certainly a simple piece of mechanism); and the compressed air is conducted alternately to one end of the cylinder and the other, thus causing the piston to strike its blows. Unless some one discovers a better way to drill rock than by the hammer-blow, it is likely that the air-drill will stand supreme for a great many years to come. No electric-drill that has thus far been devised has equalled it in simplicity; and it is certain that no electric-drill has ever equalled it in drilling-capacity, which is really the measure of efficiency.

The little air-compressor which is mounted on the hip of a

locomotive is another example of efficiency in air-apparatus. This Westinghouse pump has been pointed to as a wasteful machine; and, if it were used to drive the shafting of a shop, it would be wasteful; but when applied to the air-brake, taking its power from the boiler of the locomotive, storing it in tanks along the train, and using it at the proper time to throttle the wheels and bring to a standstill within a few hundred feet a train weighing many tons, and moving at the rate of a mile in a minute, it is the most efficient device known to mankind.

There is another little air-device, used to move switches, which is known as the electro-pneumatic system of switching; this is an admirable illustration of efficiency, and also illustrates the point to which I referred in the beginning, namely, that there is no war between electricity and compressed air, and that each has its field of usefulness. This electro-pneumatic system, employed by the great Pennsylvania railroad on its line between New York and Philadelphia, and already to be found in all large railway-terminals, represents a community of interests between compressed air and electricity. Wires and air-pipes run side by side: the one to act as a trigger, and the other as the power, to move the switch and direct the train safely, expeditiously and economically. The air-part of this system, so far as the little cylinder in which is a piston moved by air is concerned, would not be an efficient engine if applied to the common purposes for which power is used; but when doing its work on the switch of the railroad, it is one of the most efficient known. Were this not so, some other means would be employed; and we have in this instance electricity already at hand to serve the purpose, were it able to do so with equal efficiency.

W. B. CLARKE, Schenectady, N. Y.\*:—At the Albany meeting, in February, 1903, I read a paper entitled "Electrical Apparatus for Coal-Mining." This title was somewhat misleading, since a comprehensive treatment of a broad subject might have been expected; but the paper was, in fact, limited by the announcement of two kindred papers, one by Mr. H. H. Stoek on "Electricity in the Anthracite Field," and the other by Mr. R. G. Hutchins on "Electric Mine-Haulage." It was my inten-

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\* Communication received by the Secretary May 18, 1904. See Preface to this volume.

tion simply to point out the trend of recent developments in this branch of electrical engineering; and my paper accordingly described chiefly the application of high-voltage alternating-current transmission to coal-mining, and an electric mine-locomotive designed for gathering coal-cars from the room-faces. I was somewhat surprised, therefore, by the extended comments on my paper by Mr. Saunders, and I have not been able to find an earlier opportunity to reply.

Mr. Saunders remarks in his introduction that "there is no war between electricity and compressed air;" but at the same time he assumes such an aggressive attitude that some of his statements should not pass unchallenged. Fifteen years ago, compressed air had for a long time occupied this field without serious competition, and such electrical plants as were in operation were more or less experimental. Times have changed, however, and the extensive use of electrical apparatus in mining operations is an accomplished and indisputable fact.

Mr. Saunders has naturally played his best card first. The compressed-air rock-drill, although very inefficient, is a rugged machine, capable of standing the most severe abuse, and, until a comparatively recent date, has had practically no competitor. During the last three years, however, several electric rock-drills have been developed with more or less success. The principal advantage of the electric-drill lies in its much higher efficiency. An ordinary 3-in. compressed-air rock-drill indicates in the steam-cylinders of the air-compressor about 12 h.p., while an electric-drill working at the same rate requires from 3 to 4 i.h.p. The compressed-air drill has some advantage in its relative simplicity, but it attained long ago the height of its practical development; whereas the present types of electric rock-drills are of comparatively recent design, and, considering the amount of energy and capital involved, it is reasonable to expect important further improvement in them.

Mr. Saunders quotes from the Report of the Bureau of Mines of Pennsylvania for 1899 the statement that "the use of electricity in any form in coal-mines is a menace to life, limb and property."

Conservatism is certainly a commendable virtue, but just as surely there is a point where persistent conservatism becomes retrogressive. The statement thus quoted is applicable with



equal force to steam-railroads, or to the use of electricity for street-railways or for illuminating purposes, or, in fact, to a thousand and one other essentials of modern civilization. Such sweeping statements indicate merely a lack of familiarity with electrical mining-machinery; and, in view of the present extensive and successful use of electrical apparatus in coal-mining operations, Mr. Saunders's quotation has very little bearing on the present discussion.

Electric mine-locomotives are not recommended for gaseous coal-mines in which safety-lamps are exclusively employed. Mines which are "hot" to this extent are comparatively few; and in this field the mine-mule and the compressed-air locomotive may hold undisputed sway. In this connection, however it should be noted that alternating-current induction-motors are available for stationary service, in which there are no moving contacts liable to "spark;" and that the dangers of igniting an explosive body of gas are no greater with these motors than with compressed-air machinery.

Mr. Saunders quotes an extract from a paper by Mr. A. D. Adams,<sup>1</sup> entitled "The Difficulties Which Must Necessarily Be Met and Some Means of Reducing the Dangers of Faulty Insulation."

There are several wrong ways and a few right ways of installing electrical apparatus,—a statement equally true of almost every kind of machinery. Volumes have been written regarding the proper installation of air-compressors, pipe-lines and engines for mining-service. But this is not an argument against the use of compressed air. At the Albany meeting of the Institute, for instance, Dr. Ledoux presented a paper entitled "Notes on Accidents Due to Combustion Within Air-Compressors,"<sup>2</sup> in which he cited an instance where "two men were killed and four others barely escaped with their lives." Mr. Saunders says (p. 954 of this volume) that "Dr. Ledoux gives an interesting account of a serious accident, which might occur in almost any mining-plant, provided due precautions are not observed." Evidently it does not follow, in his opinion (or in mine), that the use of compressed air underground is "a menace to life, limb and property."

Mr. Saunders devotes several paragraphs to the compressed-

<sup>1</sup> *Mines and Minerals*, November, 1900.

<sup>2</sup> P. 158 of the volume.

air mine-locomotive; but it is apparent that his familiarity with this subject is not due to personal observation in the mines. In this connection, he remarks that in 1895 there were not more than six or seven compressed-air locomotives in service, "while to-day there are at least 150 of these locomotives in successful operation in mines," a record which will, he thinks, "compare favorably with that which can be shown by the manufacturers of electric mining-locomotives."

The *Report of the Pennsylvania Bureau of Mines* for 1902 is now available; and the following statistics, from the reports of 1901 and 1902, have an interesting bearing on this discussion:

*Mine-Locomotives in the Bituminous-Districts.*

	Steam.	Compressed Air.	Electric.
1901, . . . . .	134	23	231
1902, . . . . .	130	16	323

*Mine-Locomotives in the Anthracite-Districts.*

1901, . . . . .	362	51	40
1902, . . . . .	373	55	53

In the bituminous-districts during 1902 the number of steam mine-locomotives decreased 3 per cent., and that of compressed-air locomotives 44 per cent., while the number of electric mine-locomotives increased 40 per cent. In the anthracite-districts the report shows an increase in the number of steam and compressed-air mine-locomotives of 3 per cent. and 8 per cent., respectively, while the number of electric mine-locomotives increased 32 per cent. At the present writing, there are in the anthracite-districts of Pennsylvania 113 electric mine-locomotives, of which 36 are equipped with the cable-reel, and are employed for "gathering" coal-cars from the room-faces. (I may say here that at one time one of these 6.5-ton electric gathering-locomotives hauled cars from the face of a "dipping" room, where the grade was so steep that four mules working tandem were required to haul out one loaded car.)

The first electric mine-locomotive built in the United States was put in service in 1887, at the Lykens Valley colliery of the Pennsylvania Railroad Co. A year later this machine was remodeled; and it is still in every-day service, its work during 1901 amounting to 531,584 ton-miles.<sup>3</sup> The first electric mine-loco-

<sup>3</sup> R. V. Norris' discussion of Mr. Stoek's paper, p. 976 of this volume.  
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motive built by the Thomson-Houston Electric Co. was constructed in 1899 for the Erie colliery of the Hillside Coal & Iron Co.; and this machine also is still in every-day service.

The figures quoted above show that at the end of 1902 there were 376 electric mine-locomotives in the coal-mines of Pennsylvania alone. In round numbers, 450 electric mine-locomotives were built in the United States during 1902. The production during 1903 was considerably larger.

I have been specially interested in the data given by Mr. Saunders (p. 934), on the authority of Mr. E. B. Lord, of the H. K. Porter Co., of Pittsburg, well-known manufacturers of compressed-air locomotives, comparing the cost of mine-haulage with compressed-air and electric locomotives. According to Mr. Lord's figures, the cost per ton with a compressed-air locomotive amounted to \$0.01015, while the actual cost with the electric locomotive is stated to be \$0.0456, or more than four times as much as with compressed air. Figures of this nature, even when compiled on the ton-mile basis, are inconclusive and misleading, since they do not take into consideration the different conditions in the mines. In a mine where the grades are in favor of the loads, mule-haulage would very likely be cheaper than the most modern mechanical system in another mine, where severe adverse grades prevail; yet it would be ridiculous to argue the superiority of mule-haulage on the strength of such a comparison.

Mr. Lord's data regarding the cost of compressed-air haulage are taken from an able and interesting paper by Mr. J. H. Bowden,<sup>4</sup> formerly Chief Engineer of the Susquehanna Coal Company, and apply to the No. 6 Colliery at Glen Lyon, Pa. The costs of electric haulage were prepared by Mr. W. A. May, General Manager of the Hillside Coal & Iron Co., and pertain to the No. 2 Shaft at Forest City, Pa. Under date of July 1, 1903, Mr. May writes as follows:

"In *Mines and Minerals* for June, 1903, there is an article entitled "Compressed Air Against Electricity," written by W. L. Saunders, in which he compares the cost of compressed air and of electricity, using the cost of haulage at the No. 6 Colliery of the Susquehanna Coal Company and the cost of electricity at No. 2 Shaft, Forest City, Pa.

"The grade at the No. 6 Colliery of the Susquehanna Coal Company, as nearly as I can recollect, is 1 per cent. in favor of the load. At Forest City, the grade

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<sup>4</sup> *Trans.*, xxx., 566.

runs as high as 4.8 per cent. against the load; two-thirds of the coal being hauled against an average grade of 2.5 per cent. You can readily see from this why the figures seem to be in favor of compressed air. Had the article been prepared in a truly scientific manner, pains would have been taken to make the comparisons under exactly the same conditions and upon the same basis. I think the article ought to be corrected or answered, because it will unfavorably impress people who do not look deeper than the surface."

According to a published description of the compressed-air haulage-plant at the Glen Lyon colliery,<sup>5</sup> "there is an average grade of 1.07 per cent. and a maximum grade of 2.8 per cent. in favor of the load."

A few years ago one of the larger coal-mining companies in the bituminous field installed a compressed-air haulage-plant, comprising the following apparatus:—one compound-steam, three-stage air-compressor, 800 lb. pressure; three 14-ton locomotives; 5,300 ft. of 5-in. "triple strength" pipe-line; and 3,600 ft. of 2-in. "triple strength" pipe-line. The cost of this installation, exclusive of the boiler-plant, was approximately \$37,000. There were two charging-stations; and the time consumed by each trip was 45 minutes. The compressor was intended for the operation of all three locomotives; but, in actual service, there was very little margin when one locomotive was in use. Some relief could have been obtained by increasing the speed of the compressor, but for the fact that the boilers were operated at the maximum safe pressure. The cost of the maintenance of the high-pressure plant and the locomotives was respectively \$8 and \$7.50 per day. This system was employed to haul coal to a long incline, upon which the cars were delivered to the tippie by a rope-haulage.

An electric-haulage system has been installed at this mine, representing an investment of \$42,000, exclusive of the boiler-plant, and comprising two 150-kw. generators, directly connected to two 22- by 20-in. simple engines, and six 13-ton electric mine-locomotives, besides feeder and trolley-lines. One of these locomotives performs all the work of the compressed-air plant, briefly described above, making the round trip in 30 min., and is used besides on other headings for two or three hours of the day. That part of the cost of the electric plant properly chargeable, for the purpose of comparison, against the compressed-air haulage previously employed, would be about \$7,000.

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<sup>5</sup> *The Colliery Engineer and Metal-Miner*, May, 1896.

A few months before the original plant was abandoned, one of the compressed-air locomotives was "overhauled" at an expense of \$2,000.

The following is an extract from a report regarding this installation:

"We found the item of repairs very high, both on the pipe-lines and locomotives. Extremely heavy valves and fittings of all kinds had to be used, which soon became defective, due to the high pressure (800 lb.) and the character of the water and air inside the mine. We found it very difficult to keep gaskets in all places where used. . . . To use an air-locomotive in a seam of less than 6 ft. would require that roof be taken down or a locomotive of very small capacity be used. . . . We found the electric-locomotive, as applied to our work, much more flexible as well as of much higher efficiency. It was the original intention that this one compressor would take care of three locomotives instead of one, which it might have done had we been able to use steam at 130 lb. and run the compressor at 140 rev. per min., but as it was, this would not work out. We had three locomotives, one of which has long since been scrapped and still have two on hand that will no doubt soon find their way into the scrap heap, as well as the compressor. The design of the compressor is such as to preclude the possibility of using it as an ordinary low-pressure compressor, owing to its very low efficiency."

In my original paper I said that the electric motor seems to be especially adapted for the severe and peculiar conditions which attend the development and distribution of power for the operation of mining-machinery. In support of this statement, innumerable installations for various classes of service might be described; but a brief reference to a plant installed some years ago in a Colorado coal-mine will suffice. According to the report of a test made by Mr. Lewis Searing of the Denver Engineering Works Co., Denver, Colo.,<sup>6</sup> the power-plant contained three air-compressors, each with 13.5-in. high-pressure, and 22-in. low-pressure air-cylinders, and a 20-in. steam-cylinder with 24-in. stroke, supplied with steam from a battery of two horizontal 60-in. by 16-ft. tubular boilers and six plain 36-in. by 36-ft. cylinder-boilers. The mine-pump was duplex double-acting, with 18.5-in. air-cylinders, 9.25-in. water-cylinders and 12-in. stroke, and was connected to the air-reservoir at the power-plant through 3,600 ft. of 4-in. pipe-line. The report says:

"The test of the compressed-air system was made in order to obtain the total efficiency between the steam-cylinders of the air-compressors and the water-dis-

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<sup>6</sup> *Electrical World*, Nov. 25, 1896.

charge of the pump. The amount of water discharged was measured by a weir at the end of the water-discharge pipe on the surface. The weir was 16 in. wide, with a measuring stake 4 ft. up stream. A pressure-gauge was placed on the water-discharge pipe at the pump and the air-pressure was measured at the receiver. Crosby steam-engine indicators were attached to the steam-cylinders of the air-compressors.

"Observers were stationed at the various points to take readings. Observations were made through the day, commencing at 10 A.M. and ending at 5 P.M. At one time in the afternoon one minute readings were taken for ten minutes. At another time one of the compressors was shut down and the pumping done by two compressors, as a check on the indicator measurements.

"The results are somewhat surprising :

Average steam-pressure, pounds, . . . . .	80
Average air-pressure, pounds, . . . . .	50
Average revolutions per minute :	
Three compressors in operation, . . . . .	64
Two compressors in operation, . . . . .	110
Average gallons discharged by pump per minute, . . . . .	400 (3320 lb.)
Average strokes of pump per minute, . . . . .	176
Average pressure per sq. in. pumped against, pounds, . . . . .	120 (276 ft. head)
Average indicated horse-power at steam-cylinders and compressors, . . . . .	312

"As the theoretical horse-power required to force 400 gal. per minute against 120 lb. is 28, it follows that the efficiency of the system between the points measured is less than 9 per cent. It will be observed that this is the efficiency of the plant as found under actual working conditions, and includes every loss which may occur between the steam-cylinders of the compressors and the water-discharge of the pump, and that due to wear of apparatus and to inherent and other defects in the system.

"As there are many mines that are forced to employ a similar method of pumping under similar conditions, it is fair to assume that the above given efficiency is not far from the average, and that nothing short of a change of system would make a material increase in the efficiency. If we go back of the steam-cylinders to the coal-pile, we find a still more deplorable state of affairs, for, to operate these three compressors requires eight boilers, consuming 35 tons of coal per day of 24 hours.

"The probable efficiency of the proposed electric plant was estimated as follows : Direct coupled generator and engine, 80 ; line, 90 ; motor, 85 ; pump and gearing, 82 ; total efficiency, 50 per cent.

"With the above efficiency, a 60-h.p. engine would do the work now requiring 312 h.p. by the compressors. Further, assuming the installation of a compound engine, using the mine water for condensing, the total coal consumption per 24 hours would not exceed 3 tons, making a saving of 32 tons per day."

An electric power-plant was subsequently installed, containing a vertical cross-compound condensing-engine, 11 and 19 in. by 15-in. stroke, developing 150 h.p. at 250 rev. per min., direct-connected to a continuous current 550 volt generator of 105 kw. capacity, and space was provided for additional units

of the same description. Generators of this size were selected in order to have sufficient power to operate the ventilating fans, screens and shops about the mines, in addition to the regular mine pumps. The report continues :

" There are two small pumps, used as sinking pumps, which are of the duplex type of the Deane Company's make, driven through gearing by a 15-h.p. motor, mounted on the same frame. The cylinders are 7 in. by 10 in., and have a capacity of 250 gal. per min., against a head of 115 ft. The pumps are mounted on four-wheel trucks and can be moved around as desired. The station-pumps, two in number, are double-acting duplex, of the Deane Company's make, with cylinders 9.5 in. by 12 in. geared to 110-h.p. motors. These pumps each have a capacity of 600 gal. per min., against a head of 400 ft.

" Shortly after the order was given to erect the electric plant, an increased flow of water was encountered which flooded the mine, the water coming up the slope 1,500 ft. before it was held in check, and an air-pump was placed at this point to hold the water until the electric-pumps could be put in operation. The water has since been pumped down about 700 ft. by the electric-pumps, and the work has been as severe a test on the pumps as could be desired. For over a month the two small electric-pumps and an air-pump have been working side by side in an entry 10 ft. wide by 5 ft. high, with the exhaust of the air-pump (into which a stream of water is discharged to prevent it from freezing) discharging so near the electric-pump that it was necessary to protect the latter from the water by an oil-cloth. This is certainly a good test for the waterproof qualities of the motors, which are still doing their work and appear to be in excellent condition.

" The average of numerous tests on the two small electric-pumps gave the following results :

Average gallons per minute discharged,	. . .	500 (4150 lb.)
Average pressure at pumps—pounds,	. . .	56.5 (130 ft. head)
Average strokes per minute, each pump,	. . .	172
Average amperes,	. . . . .	37
Average station volts,	. . . . .	475
Average indicated horse-power at engine,	. . .	32.86
Average horse-power at pump-discharge,	. . .	16.4

" Efficiency :

Generating unit, engine and dynamo,	: . . . .	71.5 per cent.
Motor, pump and line,	. . . . .	70 per cent.
Total, from engine-cylinder to pump-discharge,	. . .	50 per cent.

" As soon as the first station pump was started and in good running order, tests were made, with the following results :

Average gallons per minute discharged,	. . .	650 (5395 lb.)
Average pressure at pumps—pounds,	. . .	64 (147 ft. head)
Average strokes per minute,	. . .	188
Average amperes,	. . . . .	53.5
Average station voltage,	. . . . .	550
Average indicated horse-power at engine,	. . .	53.6
Average horse-power at pump-discharge,	. . .	24

## “Efficiency:

Generating unit, engine and dynamo, . . . . .	73	per cent.
Motor, pump and line, . . . . .	61	per cent.
Total, from engine-cylinder to pump-discharge, . . . . .	44.5	per cent.

“Although the total efficiency with the small pumps is as high as predicted, it is nevertheless true that the engine and dynamo are working at a disadvantage with a load of but 20 per cent. of their rated capacity. In the case of the large pump, the generating unit is underloaded, as is also the pump, which combine to bring the total efficiency below the estimated 50 per cent. When the mine is finally pumped out and all the pumps and motors are running, the generator and engine will have a comfortable load, and it is safe to say that the total efficiency of the pumping plant will then be very close to 60 per cent.

“The fact remains, however, that at the efficiency above shown by the present working, the former pumping of 400 gal. per min. against 120 lb. pressure, which equals 28 h.p., can be accomplished by the electric plant with but 56 h.p. at the surface instead of 312 h.p. as before.

“The engine running the generator was supplied with steam from the battery of boilers operating the compressors, but the piping was so arranged that one horizontal tubular boiler, 60 in. by 16 ft., could be cut off from the rest of the battery and supply steam to the electric plant alone. This was not the regular practice, because all the boilers were required to supply the necessary steam, even when two compressors were running, but on several occasions, when the main steam line was shut down for an hour or more, the electric plant was run on the one boiler with ease, the safety-valve popping more than once during the run, and this with the engine running non-condensing. Actual results with the compressed-air and electric plants, which are the subject of this report, show that the work of pumping calling for 312 h.p. and eight boilers in the case of compressed air could be done by electricity with but 56 h.p. and one boiler, disregarding even the use of a compound engine and condenser.

“Further, I have no hesitancy in saying that the whole plant of pumps and machinery formerly requiring the eight boilers, taxed to their utmost, can be operated by the present electric plant with but one horizontal tubular boiler.”

The objection may be offered that the compressed-air plant described above is out-of-date; but there are many such plants in operation, and, furthermore, the electric plant would hardly be considered of high efficiency in the light of more recent developments. It follows, therefore, that the above comparison is practically true of modern compressed-air and electric-power plants designed to perform the same duty. Efficiency and economy in the operation of a coal-mine, as in other lines of business, are reflected in the balance-sheets. The progressive coal-operator admits the fallacy of the old argument that “coal costs nothing,” and realizes that the coal consumed at the mine should properly be charged at the current market-price.

Although my paper dealt only with certain applications of electricity to coal-mining, it may not be out of place to mention



the extensive and increasing use of electrical machinery in metalliferous mines, especially in the Western States and in Mexico. The difference between a successful and an unprofitable venture has been due in many instances to the employment of cheap power, obtained by electric transmission from a distant water-power or coal-mine. This is particularly true of "low-grade propositions" involving large tonnages. At one prominent mining-plant in Colorado, the cost of hoisting and milling 200 tons of ore per day by steam-power was \$218. By bringing power from an electric plant at a coal-mine 32 miles away, this cost was reduced to \$53 for the handling of 300 tons of ore per day. In the old days, the cost of operating the power-plants of the famous Comstock mines in Nevada amounted to \$34 per month per indicated horse-power. Electric power is now obtainable at a cost of from \$4 to \$7 per month per horse-power, depending upon the amount of power consumed; and the resumption of operations in the Comstock mines, which were shut down for several years, was largely due to the availability of this cheaper power.

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### Accidents Due to Combustion Within Air-Compressors.

Discussion of the Paper of Dr. Albert R. Ledoux, p. 158.

E. HILL, South Norwalk, Conn. (communication to the Secretary\*):—The phenomenon described by Dr. Ledoux, involving an apparently abnormal high temperature in the air-cylinders of compressors, has not been, so far as I know, discussed in technical literature. The common formula for the adiabatic compression of dry air does not account for heat sufficient to flash ordinarily decent cylinder-oils, nor do the text-books include in their theoretical statements any quantitative consideration of the effect of leaks in the ideal machines which they contemplate.

This lack I have endeavored to supply by preparing, in the form of a circular, for the use of the customers and correspondents of the Norwalk Iron-Works, a brief mathematical

discussion of the heat of compression in air-cylinders, the essential argument of which I here reproduce.

In the adiabatic compression of a gas,

$$\frac{T'}{T} = \left(\frac{P'}{P}\right)^{\frac{Y-1}{Y}}.$$

$T$  being the absolute temperature of the air in the cylinder just previous to the commencement of compression;  $T'$ , the absolute temperature of the discharge;  $P$  the pressure of the air before compression;  $P'$ , the pressure of the discharge, and  $Y$  a constant, peculiar to the gas. For air,  $\frac{Y-1}{Y} = 0.29$ .

If  $P$  and  $P'$  be expressed in atmospheres, and  $P = 1$ , we may substitute  $P'$  for the fraction  $\frac{P'}{P}$ .

The formula then becomes, for the compression of air from atmospheric pressure,

$$\frac{T'}{T} = P'^{0.29}. \quad (1)$$

Let  $t$  be the absolute temperature of the atmosphere from which supply is drawn.

At the beginning of the stroke the air in the cylinder has come from three sources, namely, that which was left in the cylinder clearance-spaces on the previous stroke; that which has leaked in from the discharge-valves and past the piston; and that which has been drawn in from the atmosphere.

The air which was left in the clearance had, when under compression, the temperature of the discharge,  $T'$ ; but on the retrograde movement of the piston it expands and performs work, falling in temperature to  $T$ , the temperature just previous to compression; therefore, the clearance-air need not be considered in this investigation.

The air coming from the leaks expands from the high pressure of discharge to the low intake-pressure of the cylinder, without performing work except in creating velocity in its own mass; but as it comes to rest in the cylinder, its temperature becomes  $T'$ , or that of the discharge.

The air from the atmosphere we take at  $t$ , the temperature

of the atmosphere; and in well-designed compressors this figure is correct: but in some designs the incoming air is drawn over the heated surface of piston-rod and piston, for the purpose of cooling these parts, and is thereby greatly heated, and, moreover, by restricted inlet-openings a partial vacuum is created, thereby increasing the power needed for operating the compressor, which power eventually appears in sensible heat in the air when compressed.

But this argument applies only to well-designed compressors. The air is presumed to be dry and the compression adiabatic. I am well aware that compressors universally have water-jackets, but I credit these with no cooling effect on the air during compression. The lubricant on the cylinder-walls and the thin film of air in actual contact with the jacketed surfaces may experience a slight cooling; but the mass of air is so remote from these influences as to be unaffected thereby. The presence of watery vapor in the air itself will sometimes keep down the temperature, and, more often, unskillful readings of a thermometer placed at a distance from the cylinder will give apparent results much lower than the temperatures actually existing directly at the discharge-valves.

Let  $L$  represent the amount (in percentage of cylinder-capacity) of the leaks: then

$$(1-L)t + T' L = T \quad (2)$$

the absolute temperature of the air in the cylinder previous to compression.

From equation (1) we have

$$\frac{T'}{T} = P'^{0.29} \quad T = \frac{T'}{P'^{0.29}} \quad (3)$$

From (2) and (3) we have

$$\frac{T'}{P'^{0.29}} = t(1-L) + T' L. \quad (4)$$

$$T' = \frac{t(P'^{0.29} - L P'^{0.29})}{1 - L P'^{0.29}} \quad (5)$$

Applying the formula to a single-stage compressor, at sea-level, compressing to 88 lbs. or 7 atmospheres, the atmospheric air being at 62° Fahr., we have the following temperatures of discharge, when the leaks of piston- and discharge-valves are as stated :

Leak.	Temperature.	Leak.	Temperature.
0	459° Fahr.	0.08	524° Fahr.
0.01	466 "	0.10	544 "
0.02	475 "	0.12	566 "
0.04	489 "	0.14	589 "
0.06	506 "	0.16	615 "

This, however, is not a full presentation of the case of the single air-cylinder compressor; for the reason that a very large proportion of compressors, in this country at least, are used at points high above the sea-level. Four thousand feet is a moderate assumption for an example. The barometer would be at 25.7 in., and to produce 88 lbs. gauge-pressure of this thin air requires 8 compressions. Therefore, at 4,000 ft. the same heat is developed in producing 88 lbs. as is developed in producing 8 atmospheres or 103 lbs. gauge at sea-level. We have, therefore,

4,000 ft. altitude. Gauge-pressure, 88 lbs.			
Leak.	Temperature.	Leak.	Temperature.
0	496° Fahr.	0.08	570° Fahr.
0.01	504 "	0.10	593 "
0.02	512 "	0.12	618 "
0.04	530 "	0.14	646 "
0.06	549 "	0.16	675 "

Either of these cases—at sea-level or at a higher altitude—shows the possibility of a temperature fully sufficient to produce gas from the oil-lubricant and to cause it to burn, creating excessive heat and an increased development of gas, quickly followed by explosion.

It will be seen that there is a rapid increase in heat as the leak increases. The calculation is made on leaks which are percentages of the cylinder-capacity. A leak is constant, whereas the intake of the compressor depends on the speed of the machine. It follows, therefore, that a leak of, say, 2 per cent. of the intake-capacity at 125 revolutions becomes 10 per cent. when the compressor is slowed down to 25 revolutions per minute. This quite agrees with experience. In several

cases of violent explosions brought to my notice, the compressor was running slowly at the time. The oil-feed was probably adjusted to the maximum speed of the machine, and thus was excessive for the slow speed. A larger proportional leak,—a liberal quantity of oil,—and the result is easily comprehended.

The remedy for these dangerous conditions is to have the compressor made with compound air-cylinders. Such compressors, properly proportioned, when compressing to 8 atmospheres or 103 lbs. gauge, would develop under normal conditions  $245^{\circ}$  in each cylinder.

Compound compressors are less liable to have leaks than are single cylinder-machines, because there is less difference in pressure between the discharge- and the intake-side of the pistons.

For a discussion of the problem as presented by two- and three-stage compressors, I must refer to the circular mentioned at the beginning of this contribution. I think it demonstrates that in such machines the greatest temperature due to any percentage of leak, no matter where located, is within safe limits.

But in the best modern designs of compressors, the harmful effects of leaks are entirely overcome by causing the air which leaks past the pistons to go through an inter-cooler and to be thoroughly cooled before it again enters the cylinders for compression. This keeps the temperature down; and pressures of from 3,000 to 5,000 lbs. are obtained without inconvenience or anxiety.

It should be remembered that combustion is more vigorous under higher pressure than in the open atmosphere. No doubt, there is likewise an easier oxidation and a lower flash-point for oil under pressure in the cylinder than under ordinary conditions outside. In many cases a noisome gas is generated and distributed in the mine long before the final explosion. Workmen at the front, instead of receiving pure, cool air from the exhaust of the drills or other machines, breathe a foul, stupefying, and sometimes fatal, mixture.

W. L. SAUNDERS, New York City (communication to the Secretary\*):—Dr. Ledoux gives an interesting account of a

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\* Received April 6, 1903.

serious accident which might occur in almost any mining-plant, provided due precautions are not observed. In the first place, it is very likely, though not stated, that the air-compressor in question was not a compound machine; that is, the air was not compressed in stages. This is the first point to consider when equipping a mine with an air-compressor, and when the purchaser wishes to be insured against a similar occurrence. By compressing the air in stages, that is, by the use of compound cylinders, the maximum temperature during compression is very much reduced and the liability to "flaming" is less. Having an air-compressor with compound cylinders, the next thing to do is to provide a high grade of oil to be used in the cylinders. This should be an oil with a high flashing-point and of a non-coking nature. In other words, when the oil is placed on a shovel and held over a fire there should be little or no carbon deposit left after it has been volatilized. The third point to be observed is to caution the engineer not to use much oil in the air-cylinders. A drop every five minutes is sufficient in a cylinder of ordinary size, and every engineer should feed soap-suds into the cylinders at least one day in the week, in order to wash away any deposit which may have accumulated through the use of oil which has been acted upon by highly heated air. The soap-suds may be fed through the regular oil-cup. Care should be taken not to let the machine lie idle with soap-suds remaining in it; that is, shortly before quitting-time the feeding of soap-suds should be stopped and oil-feeding substituted.

Another point is that every engineer should look into the discharge-valves, air-receiver and other places where deposits are liable to accumulate, and keep them clean. A discharge-valve which sticks, either through defective construction or through the accumulation of carbon around it, is liable to admit hot compressed air from the receiver back into the cylinder and increase to such an extent the temperature of the air to be compressed, that the flashing-point of the oil may possibly be reached. In this connection it is well to bear in mind that the intake-air should come from outside, not inside, the engine-room. This is important, because a cold intake gives a higher volumetric efficiency to the compressor, and, also, because it results in a lower maximum temperature during compression,

which means not only safety against "flaming," but economy of power. The hotter the air during compression, the greater the resistance to compression. Another reason for an outside intake is that bad air in the engine-room, or smoke or fumes from a fire, will not be sucked into the compressor and discharged into the mine.

This case also illustrates the value of inter-coolers and after-coolers in a compressed-air plant. These coolers are surface-condensers, the inter-cooler being placed between the high- and low-pressure air-cylinders, and serving to reduce the temperature between the two stages of compression. The after-cooler reduces the temperature after the last stage, and condenses and collects all such foreign matter as oil, moisture, dust, etc., sending pure, dry air down into the mine.

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### Effect of Fineness of Grain on the Fusibility of Clay.

A Discussion of the Paper of Mr. Ries, read at the Albany Meeting,  
February, 1903.

H. O. HOFMAN, Boston, Mass. (communication to the Secretary\*):—The experimental demonstration by Professor H. Ries, of the fact that a mixture of clay and finely-ground flux will melt more readily than one in which the flux is less fine and therefore less intimately mixed, is very striking, and will be appreciated especially by those who have to discuss, before classes of students, fusions on a small or a large scale. The statement, that I "rather disputed" the influence of size of flux-particles on the ease with which fire-clay mixtures fused, is an erroneous inference from the paper written by Mr. B. Stoughton and myself in 1898.<sup>1</sup> We did not undertake to study the influence upon fusibility of the sizes of the particles, either of clay or flux. What we did try to find out was, whether the common practice of finely grinding raw or burnt clay, or a mixture of the two, before testing with Seger cones for resistance to heat and to fluxes, would give the same results as if the materials were tested just as they were going to be

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\* Received June 20, 1904. See Preface to this volume. <sup>1</sup> *Trans.*, xxviii., 440.

used in making blocks or pots. We did find that fine-grinding made no difference whatever in this respect. The editor of a German technical paper,<sup>2</sup> commenting on Professor Ries's paper, says we ought to have known this before we began our experiments; but we did not know it, and, therefore, tried the experiment and put the result on record.

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### Relative Elimination of Impurities in Bessemerizing Copper-Matte.\*

A Discussion of the Paper of W. Randolph Van Liew, p. 418.

ALLAN GIBB, Mount Perry, Queensland, Australia (communication to the Secretary):—In drawing conclusions from his observations, Mr. Van Liew has apparently overlooked the reduction in weight that takes place in the content of the converter. The percentage-compositions of the matte, etc., are not directly comparable for determining the rate of elimination of the impurities; and, in the absence of exact weights (which in such an operation could not be obtained), it is necessary to determine the relation between the impurities and the copper with which they are combined at various stages, in order to get data that are comparable. Owing to a portion of the copper being removed in the slag, etc., the weight of copper in the converter is not a fixed quantity, but it is sufficiently approximate to permit very reasonable comparison.

The analytical data given by Mr. Van Liew, reduced to this basis (a ratio between the impurities and the copper with which they are combined), are shown in the table on page 958.

From this table it will be evident that Mr. Van Liew's conclusions as to the source of heat at different stages are not quite correct. During the first 10 minutes, no iron is scorified, the heat being derived from the oxidation of the sulphur, of which 7 per cent. of the total is removed in this time. From

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<sup>2</sup> *Thonindustrie-Zeitung*, Oct. 13, 1903, p. 1,875.

\* Received Nov. 11, 1903, and published without author's revision, proofs sent to him in Australia not having been returned in time for publication in this volume. Any corrections hereafter received from him will be published, either under "Errata," at the head of the index of this volume, or in the next volume of *Transactions*.



Time.	Cupola Tap.	10 Min.	20 Min.	30 Min.	40 Min. Last Skim.	70 Min. Blister-Copper.
Copper.....	Per Cent. 100.00	Per Cent. 100.00	Per Cent. 100.00	Per Cent. 100.00	Per Cent. 100.00	Per Cent. 100.00
Iron.....	46.50	46.10	31.40	16.30	3.10	.....
Sulphur.....	43.00	40.00	34.60	29.00	21.30	.....
Arsenic.....	0.22	0.18	0.14	0.12	0.11	0.0012
Antimony.....	0.28	0.24	0.18	0.20	0.17	0.006

this point till metallic copper begins to form, the heat is derived from the oxidation of both iron and sulphur. Up to this point about 93 per cent. of the iron and 50 per cent. of the sulphur are removed. From this point till the copper is ready for pouring, the heat is mainly derived from the oxidation of the remainder of the sulphur.

Mr. Van Liew's observations, together with other published matter that have come under my notice, have enabled me to compare the Bessemerization of copper-mattes with their treatment in reverberatory furnaces.

Mr. James Douglas and Mr. Edward Keller have independently published data that elucidate the Bessemer operation; but these gentlemen have confined their observations, in the main, to the original matte and the final products. The intermediate products they have examined have been those which form after metallic copper has commenced to form, and from which a proportion of the impurities has been removed by the "bottoms" produced.

The reverberatory practice which I desire to compare with Bessemerization is that in which mattes of much lower grade than white metal (75 per cent. copper) are melted without previous calcination, and roasted directly to metallic copper by atmospheric oxidation. This, to my mind, should be identical in its general effect with the Bessemer operation. Much, however, has been written, generally without exact data, to the effect that particular benefits are derived from the passage of air under pressure through the molten matte.

The following data were obtained from a charge in a reverberatory furnace worked in that way:

	1.	2.	3.	4.	5.	6.	7.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Copper.....	50.7	54.0	60.0	69.8	72.7	75.4	98.5
Iron .....	25.1	22.1	17.1	9.2	5.0	1.8	.....
Sulphur.....	23.3	22.1	22.1	20.9	20.5	20.4	.....
Arsenic.....	0.109	0.109	0.108	0.011	0.097	0.083	0.073
Antimony.....	0.036	0.038	0.042	0.042	0.042	0.038	0.022
Bismuth.....	0.023	0.022	0.024	0.025	0.020	0.015	0.008

These results, calculated to a fixed quantity of copper, are as follows :

	1.	2.	3.	4.	5.	6.	7.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Copper.....	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Iron.....	49.50	40.90	28.50	13.10	6.80	2.40	.....
Sulphur.....	45.90	40.90	36.80	30.00	28.00	27.00	.....
Arsenic.....	0.216	0.200	0.185	0.160	0.134	0.110	0.073
Antimony.....	0.072	0.072	0.070	0.063	0.058	0.051	0.022
Bismuth.....	0.043	0.042	0.040	0.037	0.028	0.020	0.008

For the purpose of comparison, I have plotted these results in curves with those of Mr. Van Liew (Fig. 1), and there is a remarkable similarity in the curves obtained. In order to make the curves clear, I have used, in plotting arsenic, antimony and bismuth, a scale one hundred times as great as that used for iron and sulphur.

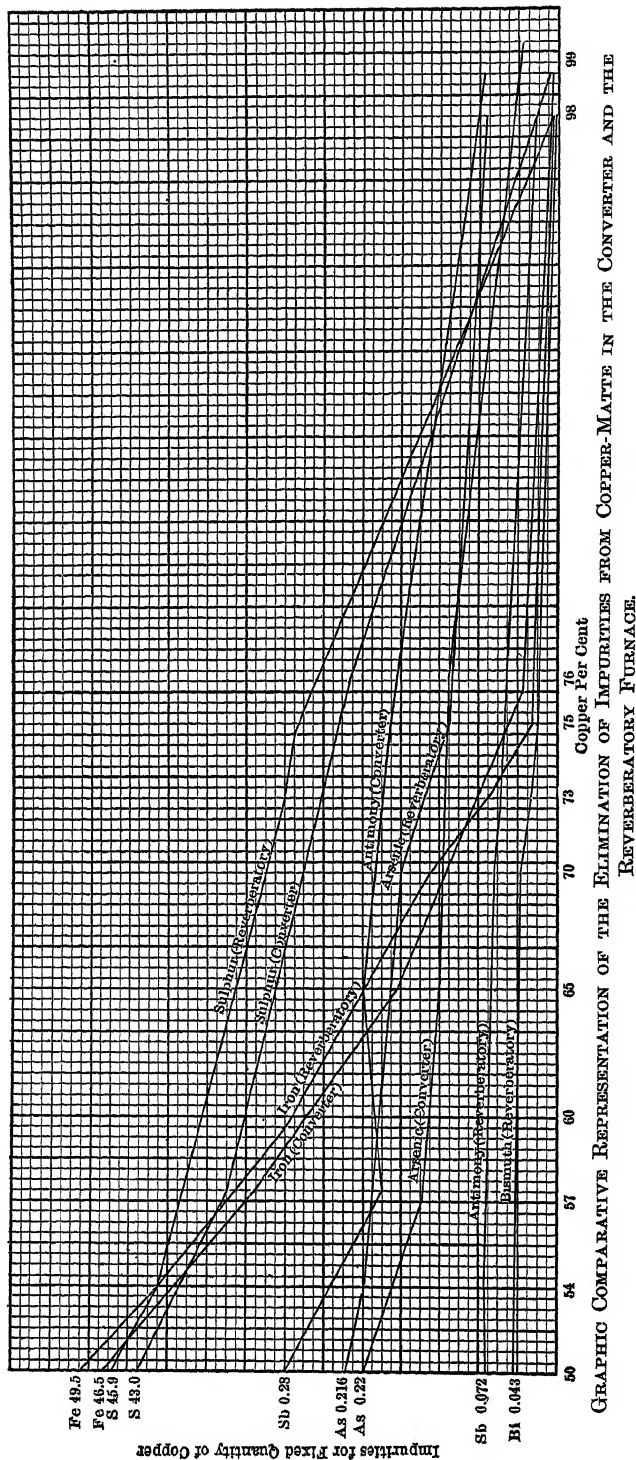
Mr. Van Liew's determinations of impurities in the copper produced are abnormal, and differ from other published data on the same subject. I have accordingly used the results of Mr. Keller for the final products.

The curves show clearly that there is little difference in the rate and degree of elimination in the two methods of treatment under consideration.

The slag from the two stages in the reverberatory furnace, viz., before and after the bulk of the iron has been removed, gave the following analytical results :

	First Stage.	Second Stage.
	Per Cent.	Per Cent.
Copper.....	7.200	39.000
Arsenic.....	0.051	0.091
Antimony.....	0.011	0.082
Bismuth .....	0.001	0.011

Fig. 1.



SECRETARY'S NOTE.—In this diagram each division of the vertical scale indicates 1 per cent. of iron or sulphur, and 0.1 per cent. of arsenic, antimony and bismuth. Each point on the respective curves indicates the percentage of the corresponding element, calculated upon the amount of copper present. Thus, when, in the converter, the bath contains 60 per cent. of copper, the amount of iron present is 28.5 per cent. of the amount of copper, or 17.1 per cent. of the total bath. The horizontal scale showing the percentage of copper is independent of the vertical scales, and does not exactly correspond with the cross-lines of the diagram.—R. W. R.

The slag from the first stage was equal in weight to the matte, and that from the second stage was about 16 per cent. of the white metal. Hence, in the first stage the impurities were distributed as follows:

	Arsenic.	Antimony.	Bismuth.
	Per Cent.	Per Cent.	Per Cent.
White metal.....	51.0	70.0	46.0
Slag.....	46.0	30.0	4.0
Volatilized.....	3.0	.....	50.0
	100.0	100.0	100.0

Again, the proportions of the total impurities remaining in the white metal were distributed as follows in the second stage:

	Arsenic.	Antimony.	Bismuth.
Blister.....	34.0	30.0	7.0
Slag.....	8.7	23.0	5.6
Volatilized.....	8.3	16.0	37.4
	51.0	69.0	50.0

The effect of Bessemerizing upon these three metals has been exhaustively studied by Mr. Keller; and I have tabulated below his summary of his observations with my own, upon the reverberatory practice, for comparison.

	Arsenic.		Antimony.		Bismuth.	
	Converter.	Reverberatory.	Converter.	Reverberatory.	Converter.	Reverberatory.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Copper.....	16.0	34.0	27.0	30.0	4.0	7.0
Slag.....	11.0	54.2	23.0	54.0	1.0	7.6
Volatilized.....	73.0	11.8	50.0	16.0	95.0	85.4
Total eliminated..	84.0	66.0	73.0	70.0	96.0	93.0

Mr. Van Liew's observations show that in the stage previous to the formation of metallic copper, 39 per cent. of the total antimony and 52 per cent. of the total arsenic are eliminated. Hence the elimination in the two stages of the converter operation is as follows:

	Arsenic.	Antimony.	Bismuth.
	Per Cent.	Per Cent.	Per Cent.
First stage.....	52	39	?
Second stage...	32	34	?
Total.....	84	73	96

From the foregoing it will be seen that there is no great difference in the degree of elimination in the two methods of treating matte. The difference in composition of the mattes under consideration will, I think, readily account for the variation in the total amounts of impurities eliminated. After having studied innumerable samples I have come to the conclusion that there is no definite law as to elimination that can be applied to copper-mattes generally. Not only does the degree of elimination of any impurity depend upon the proportion originally present, but also upon the proportion of other impurities present. There is, however, a marked difference in the manner in which the elimination is effected in the two methods of treatment.

In the case of bismuth in both operations almost the whole of the elimination is effected by volatilization.

In the cases of arsenic and antimony, however, whereas the greater part of the elimination in the converter is effected by volatilization, only a comparatively small proportion of these impurities is volatilized from the reverberatory furnace in which the greater part of the elimination is the result of scorification.

This difference of action in the two processes is not, in my opinion, directly due to the different means employed of bringing about the oxidation, but rather to the composition of the slag produced. Slag from the Bessemer operation contains a comparatively small proportion of copper. Mr. Keller gives 1.09 per cent. and Mr. Douglas gives 2.5 per cent. as an average; whereas in the first stage of the reverberatory operation the slag may contain 6 per cent. of copper, and in the latter stages it may have as much as 40 per cent. of copper.

The action of the air, whether under pressure or otherwise, upon the impurities is to form oxides. These oxides are frequently volatile per se, but also, in the cases of arsenic and antimony, have a great affinity for cuprous oxide. Hence, in

the presence of slag containing considerable proportions of cuprous oxide, these two elements, in part, enter the slag in the form of arsenates and antimonates of copper, compounds that are not volatile at the temperature of the operation. Similar compounds of copper and bismuth are not formed; hence this element as far as it is oxidized is removed by volatilization, whatever the composition of the slag.

The larger proportion of copper scorified in the reverberatory practice is certainly the outcome of the longer exposure to the oxidizing agent, and the difference of action may, to that extent, be said to be due to the difference of air pressure, but I do not consider that we have any data to show that high pressure air can, in itself, affect the impurities differently from air at atmospheric pressure.

Mr. Keller has advanced the opinion that air blown through the copper as it is produced may have a purifying effect. Certainly blowing through copper from which practically all the iron and sulphur have been removed (that is, overblowing) would have such an effect; and I am inclined to think that this will account for the abnormally low proportions of impurities that Mr. Van Liew found in the bar-copper. But that such purification can take place, at least to any great extent, when sulphides are present, can hardly be possible in view of the possibility of concentrating the impurities in "bottoms" in the David "selecteur." In this apparatus air is introduced through bottom-tuyeres, and would have the fullest effect on the copper produced. I had an opportunity of seeing this apparatus worked at the D'Eguilles works of the Société des Cuivres de France, and the samples that the manager, M. Vuigner, allowed me to take gave me the following data:

	Copper.	Arsenic.	Antimony.
	Per Cent.	Per Cent.	Per Cent.
Matte from cupola....	36.3	0.41	0.52
Bottoms.....	92.6	2.08	5.26

These data represent a concentration of 70 per cent. of the total arsenic, and 72 per cent. of the total antimony in the metallic "bottoms"—proportions not greatly differing from those usually obtained in the best selecting operation, as carried out in reverberatory furnaces.

## The Assay of Zinc-Box Residues from the Cyanide Process.

Discussion of the Paper of Prof. Richard W. Lodge, p. 432.

CHARLES H. FULTON, Rapid City, So. Dak. (communication to the Secretary\*): Professor Lodge takes issue with Mr. Crawford and myself on results obtained by the scorification-method of assay on "zinc-box precipitates," which we published in a paper on the subject in the publication named below.<sup>1</sup> In our work we found that the normal scorification-method, employing only test-lead and a very little borax-glass as a flux, gave results on cyanide precipitates which were invariably very low, loss having taken place by slag-absorption and volatilization, as repeated slag-assays failed to bring up results similar to those obtained by other methods.

Professor Lodge and his students made a large number of what he terms "scorification-assays," but which, I would like to point out, have claim to that name only in that they are made in a scorifier, for the charge consists of test-lead, a very considerable quantity of borax-glass (10 grams), at times also of litharge and even charcoal, so that the assays are in reality crucible assays. A scorification-slag proper is an oxide-slag to all intents and purposes, while the slags made by Professor Lodge are borates, certainly acid-slags.

It has been our experience since our paper, *The Assay of Cyanide Precipitates*, was published, that the nature of the assay slag in the crucible assay has a great influence on the result. A "basic" or oxide-slag invariably causes high slag-losses, while an "acid" slag gives very fair results on material containing much zinc. This has also been the experience of a well-known cyanide chemist with whom I have corresponded on the subject, who found that acid crucible-slags gave good results on zinciferous material, but that the scorification-method, as ordinarily employed, gave high slag-losses.

I believe that if the conditions are such that the zinc can be

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\* Received January 18, 1904. See Preface to this volume.

<sup>1</sup> *School of Mines Quarterly*, xxii., No. 2, pp. 153 to 162, January, 1901.

carried into the slag as a silicate or borate (and it is fairly well established among metallurgists that in lead-slugs the zinc exists in the main as a silicate), good results can be obtained by the crucible-method. Professor Lodge's crucible-slugs were all basic, which I think accounts for his low results by that method.

The zinc in cyanide precipitates exists in the form of metallic zinc and zinc oxide. In the crucible-assay it is very desirable to oxidize this metallic zinc in order to insure its slagging off. This is usually done by having considerable litharge present, allowing metallic zinc to reduce lead, and adding just sufficient reducing agent to give enough more lead to make a proper-sized button. Usually, smelting-plants which purchase cyanide precipitates make no allowance for slag-absorption, allowing only a cupel-correction, so that it is very desirable to produce as clean a slag in the first assay as possible.

Professor Lodge's material contained comparatively little zinc (but 14.3 per cent.), while the material our paper was written on contained 42.3 per cent., or three times as much, which fact would most certainly have an influence on the assay.

We have also found that ordinary analytical balances, such as are used in quantitative chemical work, are amply accurate to weigh out the assay sample, weighing as they do to 0.1 milligram.

The error of 0.1 mg., in weighing a sample of Professor Lodge's material, would make a difference of 0.016 mg. in the weight of the resultant gold bead, and if 0.05 assay tons are taken for assay this makes a difference of \$6.60 in value per ton of precipitates, which, when one considers that the amount shipped by a mill is rarely more than 100 to 150 lb., is insignificant.

There is no question that it is very difficult to obtain an average sample of zinc precipitates, and the finer the screen through which the material is put, the better. Professor Lodge's 200-mesh bolting cloth is not too fine.

Regarding the accuracy of the "combination wet and dry" methods and the crucible-method, the following are results recently obtained on cyanide precipitates:



Nitric Acid Method.		Sulphuric Acid Method.		Crucible Method.	
Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
Oz.	Oz.	Oz.	Oz.	Oz.	Oz.
1012.2	21.4	1042.6	22.1	1063.5	22.6
1030.2	17.9	1064.1	22.5	1075.5	23.4
1025.9	21.0	.....	.....	.....	.....
992.1	17.3	.....	.....	.....	.....

In the nitric acid method the gold was recovered from the solution by precipitation with sulphuretted hydrogen, after the removal of excess acid, showing loss from 0.25 to 3.65 oz. gold per ton in the solution.

These results from the solution are not anywhere near as high as those obtained by us and recorded in our paper already referred to.<sup>2</sup>

We now believe the large amounts of gold obtained from the nitric acid solutions at that time to have been due to some substance contained in the precipitates, whose exhaustion, unfortunately, precluded further investigation regarding the nature of this material.

In reference to the "Whitehead method" giving low results on silver, it has been often demonstrated<sup>3</sup> that it and similar methods give good results on silver, but are low on gold, owing to the solution of gold. Professor Lodge ascribes his low results on silver to a solution of the precipitated silver bromide by a too strong solution of potassium bromide in excess. This will certainly not occur if the so-called "normal" salt solution is used in approximately the proper amount to precipitate the silver present.

The combination methods require great care in the handling of the precipitated silver chloride, to prevent mechanical loss and loss by volatilization.

I differ with Professor Lodge regarding his statement that an excess of litharge causes increased slag absorption of silver or gold; his experience in that respect being different from that of many assayers. In a recent paper in the *Transactions*,<sup>4</sup> Professor E. H. Miller comes out as a strong advocate for "oxide-slugs," mainly excess litharge and soda; and Mr. Perkins<sup>5</sup> in a recent

<sup>2</sup> *Ibid.*

<sup>4</sup> *Trans.*, xxxiv., 387.

<sup>3</sup> *Trans.*, xxx., 1121.

<sup>5</sup> *Trans.*, xxxi., 113.

paper describes a valuable "excess litharge method," which I have found to give excellent results on very impure ores.

PROF. RICHARD W. LODGE, Boston, Mass. (communication to the Secretary\*): In regard to the exception which Mr. Fulton takes to my term "scorification-assays," I have always understood that this term applies to an assay made in a scorifier with test-lead, or with test-lead and borax. Some ores can be decomposed with lead alone, others need a little borax-glass and others a great deal. In the published results, the most satisfactory method was found to require 60 g. lead and 10 g. borax-glass. If in a scorifier a correct amount of borax-glass is used in place of an incorrect one, does this make it any less a scorification-assay? Many ores other than zinciferous, and some base-bullions, require a large amount of borax-glass to decompose them. In the assay of mattes rich in copper, borax-glass and silica are both necessary, the latter especially so after the first-scorification. In some cases a little soda is used in the scorifier. Does the addition of these substances make the assay less a scorification?

If no test-lead had been used in the scorifier, but the lead had been reduced from litharge, then I would say the objection was a good one.

In regard to the excess of litharge, for the last 20 years two crucible-methods for assaying sulphide ores have been taught to students at the Massachusetts Institute of Technology:

1. The "iron method," in which high soda, always at least twice the ore, low litharge and an excess of iron are used;
2. The "litharge method," in which we always use the same amount of soda as of ore, high litharge (from 150 to 175 g. per 0.5 assay ton of ore in some cases) silica and niter if necessary.

The second method often gives low results for silver which may be accounted for by the niter where much is used. In other cases, comparative tests on ores of about every character lead me to believe that the low results are due to the high litharge, especially where the ores or products are rich in silver.

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\* Received February 5, 1904. See Preface to this volume.

## Geological Relations of the Manganese Ore-Deposits of Georgia.

Discussion of the Paper of Thomas L. Watson, p. 207.

CHARLES CATLETT, Staunton, Virginia (communication to the Secretary\*): I have had occasion to examine a few of the iron-deposits in the immediate section of which Mr. Watson speaks, but have not had an opportunity to visit any of the well-known manganese-deposits. I find the former can be very safely read in light of the information which it is possible to get from the same measures farther north, where they are less completely decomposed. In the same way all of his conclusions in regard to the geology of the manganese-deposits bear out their occurrence in Virginia, when we allow for the far more complete decomposition in the section he has so carefully studied. Judging from the same measures farther north, I think he will be safe in agreeing even further with Penrose as to the origin of these ores. Some years ago I carried on prospecting work in these measures in Virginia, in which the iron-ore and manganese-ore were followed down to undecomposed material, and in which their occurrence along a definite line of stratification, and apparently a part of the original bedding, was plainly shown. This is unusual, however, even in the section of which I speak. At Crimora, Virginia, the decomposition is quite complete; at the same time the results of some hydraulic work, which has been done there, disclosed, at the time of my last visit, the manganese interbedded in the residual clay along a definite and distinct plane parallel to the bedding of the underlying quartzite, and which, as is shown at other points, corresponds with the plane of the original material.

In view of the phosphorus reported by Mr. Watson as existing in the Cartersville ores, and the not uncommonly prohibitive amount in southern ores, I am interested to know whether

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\* Received October 16, 1903.

the percentage he speaks of as "low" is not still sufficient to affect seriously their market value; and whether its distribution is such as to make one satisfied, in opening a deposit in that section, that the phosphorus-content can reasonably be depended on; and whether, as a rule, the surface samples are not apt to be lower in phosphorus than those at a greater depth. I speak more especially of the ores more closely associated with the Weisner quartzite. I had occasion some years ago to examine a manganiferous iron-ore property, in this geological horizon, from which between 3,000 and 4,000 tons of ore had been shipped, all of which ran less than 0.1 per cent. in phosphorus, or what would, I believe, under the scale of the steel companies, be considered low phosphorus. On investigating the deposit at a greater depth, however, it was found that the phosphorus rose to between 0.2 and 0.3 per cent. This fact led to my investigating with some interest the question whether the phosphorus had been leached out; and it was found that in those particular surface ores such was undoubtedly the case, and, further, that in many ores from this horizon the phosphorus could be very greatly reduced by crushing and roasting the ores (so as to make them more porous) and leaching with water. In one case as much as 50 per cent. of the phosphorus was removed by one treatment. It is not improbable that additional value might in this way be given to deposits which are now unmarketable.

Some of the iron-ores which are elsewhere applicable to the geological formation of the Cave Spring section undoubtedly previously existed as sulphides, and are quite frequently encountered in a partially oxidized condition. I have never encountered any manganese existing in sulphide, though it seems not improbable that it may have been formed as such. I would point out that the relative commercial value of a deposit formed by the decomposition of a rock in place, which contained the mineral more or less widely disseminated, is very much affected by the inclination of the rock to the surface. When the rock lies at a comparatively slight inclination to the horizontal and, at the same time, conforms closely to the contour of the ground, the total amount of valuable material is apt to be greater, and the relative proportion of dead or waste material is apt to be less than when the contrary is the case.

THOMAS L. WATSON (communication to the Secretary\*): The widespread occurrence of iron-ores in the Georgia manganese region studied by me<sup>1</sup> has been the subject of recent study by Doctor C. W. Hayes, of the U. S. Geological Survey,<sup>2</sup> and by Mr. S. W. McCallie, of the Geological Survey of Georgia.<sup>3</sup>

The genesis of the iron-ores is set forth in considerable detail in the above publications, by both Hayes and McCallie, and with essential agreement on all but one type of the deposits. The concentration-deposits of Hayes, in the Cartersville district, are referred by McCallie to derivation from iron carbonate. The latter cites the occurrence of a stratum of iron carbonate 20 and more feet in thickness in the recent developments of the Sugar Hill district to the north of Cartersville, and further refers to the probability of a part of these iron-ores having been derived to some degree, at least, from the oxidation of the seemingly large amount of pyrite disseminated through the so-called slates and quartzites. Both Hayes and McCallie agree as to the genesis of the Cave Spring deposits and the adjacent ones, but they do not ascribe their origin to oxidation from pyrite. It seems exceedingly doubtful, therefore, that any of the workable ores in the Georgia area, so far studied, can be referred to the direct oxidation of pyrite.

While the iron-ores are somewhat more or less closely associated with those of manganese, in the Georgia area, I gave them little concern for the reason that they had only, within a few months, been studied and reported on in much detail. However, at no point over the area, where study was devoted by me to the more closely related deposits of iron-ores to those of manganese, was I convinced that they were derived in any degree from the oxidation of pyrite.

Concerning the question raised by Mr. Catlett, whether the "low" phosphorus-content in the Georgia manganese-ores, stated in my paper, is not sufficient to affect seriously their market value, I can only say that from the records of hundreds of the car-load lots of the ore shipped to the principal steel-making plants in the United States, comparatively seldom has

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\* Received October 22, 1903.

<sup>1</sup> *Transactions*, xxxiv., p. 207.

<sup>2</sup> *Transactions*, xxxi., pp. 403-419.

<sup>3</sup> *Bulletin No. 10-A*, Geological Survey of Georgia, 1900.

any reduction been made for phosphorus. There are, in the Georgia region, undoubted deposits of manganese of which this cannot be said, for the reason that the phosphorus ranges considerably above the limit in this constituent generally allowed by the steel companies. More often is the price paid by the companies for the Georgia ores at the furnaces lowered on account of high silica, rather than of high phosphorus. Even this trouble might have in many cases been saved had the ores been properly cleansed previous to marketing them. From the very large number of commercial analyses of the ores accessible to me and the price paid for them, I feel reasonably safe in saying that the phosphorus-content, as a rule, can be depended upon, in the Georgia ores.

Whether the surface ores are apt to be lower in phosphorus than those at greater depths, I have not the data at hand to say. In the absence of this data, I am strongly of the opinion that there is or should be no appreciable difference in this constituent for different depths, for the reason that the workable ores, from the nature of their occurrence and genesis, are without exception entirely limited to the residual clays derived from the underlying rocks. No considerable depths are ever attained in their working, the deepest mines, of which there are only a few, rarely exceed 100 ft., and hence the ores are all, so far as developments extend, surface ores comparatively speaking. It may be possible, and even probable, that the loose scattered fragments of the ore (float) over the surface, subjected to exposures of long continued atmospheric influences, may show some appreciable difference in the phosphorus-content and prove to be some lower in this constituent than the buried ores.

Finally, I will say that from my own observations and study of the Georgia area, the manganese-ores are not to be considered in any sense interbedded deposits, but are embedded ores. My close study of the area has entirely failed to indicate the occurrence of manganese in the fresh rock, at any point, in anything more than a trace, if at all. Not having sufficient personal familiarity with the Virginia ore occurrences mentioned by Mr. Catlett, I am not entitled to any expression of opinion as to whether the Georgia ores can be interpreted in light of the similar Virginia ores.

In connection with the question of the effect of phosphorus on the market value of manganese-ore, the following schedule, issued by the Carnegie Steel Co. under date of February 1, 1903, is of interest. I understand that this company is not making any purchases of manganese-ore at the present time, February, 1904.

*Schedule of Prices, February, 1904, Which Will be Paid by the Carnegie Steel Co., Pittsburg, Pa., Per Ton of 2,240 Lb. for Domestic Manganese-Ore Delivered at the Lucey Furnaces, Pittsburg, Pa., or at the Edgar Thomson Furnaces, Bessemer, Pa.*

	Metallic Manganese.	Prices per Unit.	
		Iron.	Manganese.
	Per Cent.		
Ore containing above.....	49	5 c.	25 c.
Ore containing from.....	46 to 49	5 c.	24 c.
Ore containing from.....	43 to 46	5 c.	23 c.
Ore containing from.....	40 to 43	5 c.	22 c.

NOTE.—Ore containing less than 40 per cent. manganese, and more than 10 per cent. silica, or 0.15 per cent. phosphorus, is subject to acceptance or refusal at the buyer's option.

Prices are based on ores containing not more than 8 per cent. silica and not more than 0.10 per cent. phosphorus, and are subject to deductions as follows:—

For each 1 per cent. in excess of 8 per cent. silica there shall be a deduction of 15c. a ton; for each 0.02 per cent., or fraction thereof, in excess of 0.1 per cent. phosphorus, there shall be a deduction of 1c. per unit of manganese per ton.

Settlements are based on analysis of sample dried at 212° Fahr. The percentage of moisture in the sample as taken, being deducted from the weight.

Prices subject to change without notice, unless otherwise specially agreed upon.

The ores to which I specifically referred were also limited to the residual clays, though lying along a plane distinctly paralleling the underlying quartzite, which was there very slightly decomposed. The fact that it was possible in some cases to re-

duce the phosphorus as much as 50 per cent. by heating to a red heat and treating with water for a space of about two hours would indicate that, in some cases at least, a portion of the phosphorus was in a form to be easily leached out. I am not prepared to say what changes may take place immediately at the surface which would aid in removing a portion of the phosphorus under ordinary atmospheric conditions, though I feel confident that such is not uncommonly the case. Is it not possible that the formation of more or less vegetable acid may have some effect?

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### Observations on Mother-Lode Gold-Deposits, California.

A Discussion of the Paper of W. A. Prichard, p. 454.

H. W. TURNER, San Francisco, Cal. (communication to the Secretary\*): This excellent paper apparently represents the results of extensive observation and experience among the mines of the Mother-Lode, but I strongly question its relegation of the dioritic intrusions to the period of fissuring and ore-deposition.

The dioritic rocks referred to are evidently those areas represented by Ransome in the Mother-Lode folio<sup>1</sup> as meta-diorites, and, as he says, are merely basic *facies* of the great granodiorite batholite of the Sierra Nevada. If followed downward I venture to predict that these meta-diorite areas would be found to connect with the great batholite, and to be in fact mere apophyses of it. I see no reason for supposing that these apophyses were intruded into the Mother-Lode slates at a period later than the intrusion of the granodiorite in general.

The period of fissuring was subsequent to this intrusion; the basic granodiorite or quartz-diorite areas of the foot-hill region are themselves fissured extensively; and the fissures were filled with quartz, apparently during the same general period of the formation of the Mother-Lode veins.

Prichard and Ransome<sup>2</sup> both refer to the deposition of quartz in the same lode at different periods resulting from a reopening of the fissures. Some time since, I saw in the stopes of the

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\* Received October 5, 1903.

<sup>1</sup> *Mother-Lode Folio*, U. S. Geological Survey.

<sup>2</sup> *Op. cit.*



Gwin mine what seemed a very plain case of two periods of vein-filling.

The main mass of the lode, quite wide at this point, was composed of crushed slate and quartz, the latter in the form of boulders, angular fragments, and lenses of quartz lying at various angles in the crushed slate. Evidently the original vein had been ground up by movements subsequent to its formation.

Forming a more definite vein in the crushed material was a band of ribbon-quartz, lying parallel to the plane of the vein, and exhibiting no evidence of crushing and distortion, except that the ribbon structure itself may be due to shearing stresses, a point which has not yet been determined. I supposed this ribbon-quartz to represent a later deposition, in a fissure formed after the period of crushing. Mr. J. W. C. Maxwell informs me that, many years ago, he observed the same thing in the Amador Consolidated mine at Sutter Creek.

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### The Garnet-Formations of the Chillagoe Copper-Field, North Queensland, Australia.

Discussion of the Paper of George Smith, p. 467.

H. W. TURNER, San Francisco, Cal. (communication to the Secretary\*): The recent papers in the *Transactions* by Vogt,<sup>1</sup> Lindgren<sup>2</sup> and Weed,<sup>3</sup> on ore-deposits that have formed as a direct result of igneous intrusions, show that deposits of this class are widely distributed over the world, and are occasionally of commercial value. This is especially true if we consider under this head all deposits in the zone of contact-metamorphism, the origin of which may be ascribed to the gases and vapors generated by the heat of the intrusive rock.

Mr. Smith's description of the Chillagoe garnet-formations containing copper, leads one to regard these deposits as of contact-metamorphic origin, although he gives certain reasons for supposing them to be of igneous origin.

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\* Received August 26, 1903.

<sup>1</sup> *Trans.*, xxxi., 137-140.

<sup>2</sup> *Trans.*, xxxi., 226-244.

<sup>3</sup> *Trans.*, xxxii.

The occurrence of calcium silicates (wollastonite, epidote, garnet, pyroxene, etc.), along the contact of intrusive igneous rocks with calcareous sedimentary rocks, is generally assumed, with good reason, to be due to contact-metamorphism. It is true that when granitic rocks are intruded into pure limestones the result can only be the recrystallization of the calcium carbonate into calcite, unless silica, alumina, etc., are supplied from another source. The analyses of the Chillagoe limestone given by Mr. Smith show that it is nearly pure calcium carbonate. Hence, in order to produce garnet, which ordinarily contains lime and silica and iron as essential constituents, the source of the silica and iron must also be accounted for. It would not be an extreme view to suppose that these elements were supplied by the thermal waters, vapors and gases emanating from the intrusive magma during cooling, especially as the magma itself contains them. Mr. Smith's reasons for suggesting an igneous origin for the garnet-formation appear to be:—

1. That there seems to be no transition between the garnetiferous rocks which contain the ore and the limestone adjoining.

2. That the garnet-rock has been determined as originally composed of bronzite, which has altered to bastite and garnet, forming an eclogite, understood by him to be an igneous rock.

Now, since bronzite is a magnesium-iron silicate, it can readily alter to bastite, which is a hydrous magnesium silicate with usually some iron, but it cannot form garnet without the addition of alumina with some base, which would be calcium, if it is a lime-garnet.

Garnet as an original crystallization from an igneous magma is common, as in many granites. Possibly the rock in question was originally bronzite and garnet. The latter mineral is found in diallage rocks<sup>4</sup> in Europe, in serpentine in South Africa, and elsewhere; and these rocks are supposed to be of igneous origin. In most occurrences of this character, however, the garnet is pyrope, a magnesium-aluminum silicate, and not a calcium silicate. It would be interesting to have a thorough description of the Chillagoe eclogite, with a determination of the nature of the garnet; and it is to be hoped that Mr. Moulden, cited by Mr. Smith, may furnish this. Mr. Smith

<sup>4</sup> Rosenbusch, *Mikroskopische Physiographie der Massigen Gesteine*, 1896, p. 369.

speaks of eclogites as being of igneous origin; but Rosenbusch classes most of them with the lime-silicate contact-metamorphic rocks.

In view of the facts brought out in the recent papers of Vogt, Lindgren and Weed, cited above, it appears probable that the Chillagoe deposits are of contact-metamorphic origin. According to Mr. Smith, the frequent association of original oxides and sulphides, said to be common in contact-metamorphic deposits, is one of the features of the Chillagoe field.

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### Application of Electricity in the Anthracite Coal-Field of Pennsylvania, with Special Reference to the Wyoming Field.

Discussion of the Paper of H. H. Stoek and G. W. Harris, p. 512.

R. V. NORRIS, Wilkes-Barre, Pa. (communication to the Secretary\*): The electric-haulage plant of the Short Mountain Colliery of the Lykens Valley Coal Co., mentioned by Messrs. Stoek and Harris as the earliest underground electric plant in the United States (and, I believe, the second in the world), was started July 23, 1887. The plant, designed by W. M. Schlesinger, consists of two motor-cars, each weighing about 5 tons and equipped with a 32-h.p. Union Electric Co. motor. The original general construction of the motor-cars is shown by Mr. Stoek; but they were rebuilt, the same year, with some important improvements in design; and, except for strengthening the frames and replacing the original pig-iron weights by a casting, they have not been greatly changed since that time.

The generating-plant consists of two Union Electric Co. dynamos of 65-h.p. each, driven by an old 18- by 48-in. plain slide-valve engine; the ordinary voltage is 450 and the amperage varies from 40 to 200; the average indicated h.p. of the engine is about 75, with a steam-consumption of about 75 lb. per i.h.p. per hour; and as the cost of steam at this colliery is 8.11 cents per 1,000 lb., the steam-cost per day is about \$4.56.

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\* Received February 15, 1903.

The two haulage-lines are in No. 1 drift (9,500 ft. long), and Bear Gap Tunnel (10,400 ft.).

The ordinary train is 15 mine-cars, weighing (empty) 1 ton each, and carrying 2.25 tons of coal, or 3 tons of rock. The work during the year 1901 was—

	No. 1 Drift.	Bear Gap Tunnel.	Total.
Cars of coal.....	48,120	14,681	62,801
Cars of rock.....	4,233	235	4,468
Tons of coal.....	108,270	33,032	141,302
Tons of rock.....	12,699	675	13,374
Total tons.....	120,969	33,707	154,676
Ton-mileage, effective load.....	217,744	66,600	284,344
Ton-mileage, dead load, cars and empty return-trips..	188,471	58,769	247,240
Gross ton-mileage.....	406,215	125,369	531,584

The total cost of labor and repairs for the year 1901 was \$4,510.42, and the approximate cost of power as noted above was \$1,286.83, making the total operating expense \$5,797.25, or, per ton-mile, net work:—labor and repairs, \$0.0159; power, \$0.0046; total, \$0.0205. Per ton-mile, gross work:—labor and repairs, \$0.0086; power, \$0.0013; total, \$0.0109. As compared with the cost for the years 1897 and 1898 of compressed-air motor-haulage at No. 6 Colliery of the Susquehanna Coal Co., given in Mr. J. H. Bowden's paper:<sup>1</sup>

	1897.	1898.
Per ton-mile, net work.....	\$0.0189	\$0.0193
Per ton-mile, gross work...	\$0.0105	\$0.0107

Mr. Bowden's costs include interest and depreciation, and the power-cost was based on the cost of steam in the Wyoming region. To make a proper comparison with the above-noted cost of motor-haulage, the figures should be put on the same basis, which gives the comparative average costs for the two years as follows (the power-cost being taken at \$0.0811 per 1,000 lb. steam for the 52,200 lb. stated to be used per day of 10 hours,

<sup>1</sup> *Trans.*, xxx., 566.

interest and depreciation deducted): Per ton-mile, net work:—labor and repairs, \$0.0084; power, \$0.0040; total, \$0.0124. Per ton-mile, gross work:—labor and repairs, \$0.0048; power, \$0.0022; total, \$0.0070.

It should be noted, however, that the compressor is stated to use only 35 lb. steam per 1 h.p. per hour, while the electric-plant engine uses 75 lb., which would raise the average comparative cost of air-haulage to \$0.0169 net, and to \$0.0095 per gross ton-mile.

The working-time for the air-motors was 179 days in 1897 and 160 days in 1898, while the working-time for the electric motors was 282.2 days in 1901.

The average operating-cost of mule-haulage, given in the paper cited above, deducting interest and depreciation, was:—Per ton-mile, net work, \$0.049; per ton-mile, gross work, \$0.027.

A summary of comparative average costs of operation, labor, repairs, supplies and power is as follows:—

	Electric Motors.	Air- Motors.	Mule- Haulage.
Per ton-mile, net work.....	\$0.0205	\$0.0124	\$0.049
Per ton-mile, gross work.....	\$0.0109	\$0.0070	\$0.027

Of course, the electric and the compressed-air motors, both being pioneer plants in the anthracite-region, were working in different regions and under different conditions; the length of run of the two compressed-air motors being but 4,000 and 2,100 ft. respectively, and the average number of cars hauled per year being 111,124. Moreover, the labor conditions were very much more favorable to economical work in 1897 and 1898 than in 1901. And, finally, it must be remembered that the electric-haulage is of the type of 1887.

### The Condition and Action of Carbon on Steel.

Discussion of the Paper by Mr. Herbert E. Field, p. 559.

WILLIAM METCALF, Pittsburg, Pa. (communication to the Secretary\*): I am greatly pleased that some one has at last reached what I have long regarded as the only rational explanation yet given of the reasons why steel hardens.

Mr. Field claims originality; and doubtless he is entitled to it, but not to priority, which must be granted to Prof. John W. Langley.<sup>1</sup>

The difference between them is that Langley says "Tension, probably molecular," and Field says "Tension, probably atomic"; in either case the vital thing after solution is tension.

Mr. Field's argument carries with it the idea of increase of density with hardness; if he will study the facts farther he will observe that the case is just the reverse; that is, the greater the hardness the less the specific gravity; it was just this fact that led Langley to his conclusion, in which I have always concurred.

Field agrees with Langley that the first requisite for perfect hardening is complete solution of the carbon, not combination; and all of the facts agree with this conclusion.

The fact that increase of hardness causes reduction of density was first brought to my notice some 40 years ago, when Major William Wade was testing Jones & Laughlin's cold-rolled iron, and I was his assistant. We found the well-known increase of tenacity, compressive strength, torsional resistance, and hardness. Specific-gravity tests showed that the cold-rolled iron was less dense than the hot-rolled bars.

Major Wade was so surprised at this result that he gave me a lot of samples to weigh, saying that he was not satisfied with

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\* Received October 13, 1903.

<sup>1</sup> *Transactions of the American Society of Civil Engineers*, xxvii., p. 403, and *Steel: A Manual for Steel Users*, by William Metcalf, p. 106, on hardening, and pp. 149 and 150, on solution of carbon, and tension.

his work, but vouchsafing no reason. My results agreed with his; and he then told me his difficulty in reconciling the facts to the general theory existing at that time.

Not yet satisfied he went to the mill and measured carefully a pickled hot-rolled bar and calculated its cubic content; he then had the bar cold-rolled, and on remeasuring it he found that the increase in length more than compensated for the reduction in diameter—a result which established the case.

Langley's beautiful experiments on specific gravities developed the fact that, as far as we know, increase of hardness is always accompanied by decrease of density; and it is this fact that makes both his and Field's theory of tension so reasonable and altogether satisfactory.

Modern metallographers have confirmed nearly every conclusion that we life-long metallographers—shop men who must study and use metallography every day—have reached and published over and over again, except the solution and tension theory of hardening, which is so beautiful and in such complete accordance with facts.

We can hardly criticise them for this, as it is not to be expected that they can see individual molecules, or atoms, with the best of microscopes. I am glad to see in Mr. Field one man, after Langley, who has mental clearness of vision.

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### The Sperry Vanning-Buddle.

A Continuation of the Paper of the same author, p. 572.

EDWIN A. SPERRY, Denver, Colo. (communication to the Secretary\*):—Since writing my former paper on this subject I have had several discussions concerning some of the statements contained therein, with men who have had a large experience with the old form of buddle, both in this country and in Cornwall. The present paper gives a brief statement of some of the points debated, together with the arguments and proofs in their substantiation.

The first point is the question of the capacity. In my origi-

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\* Received March 25, 1904. See Preface to this volume.

nal paper I stated that with a 10-ft. table we developed a capacity of more than 20 tons in 24 hours. This seemed to be almost incredible, for the reason that a buddle of the same size would hardly give one-fifth of that quantity. It was admitted, however, that the addition of hammers at the edge of the buddle increased the capacity and improved the efficiency, and it is evident that a similar and better result is acquired by the addition of the circular vanning motion, a fact which was proven by actual results.

Referring to the plan of the top of the table (Fig. 3), it will be seen that the pulp is gradually added to the surface from the initial line of feed to the final line, and that by the vanning motion it is progressively stratified at all points. The effect of this is, that at the point where the pulp is first subjected to the action of the wash-water, it is in a condition requiring only the washing off of the silica, a circumstance which allows the greatest possible load and avoids the necessity of relying on the sprays to accomplish the stratification of the pulp, thus leaving the washing action solely, for the spray.

Theoretically, the capacity of the table is limited only by the quantity of the metallics which comes in direct contact with the surface, and is affected by the clinging action of the rubber. Basing the calculation on this principle, and taking for illustration an ore that will concentrate to 10 per cent. or 10 tons into one, the following results are obtained: A table 10 ft. in diameter has an area of 78 sq. ft., of which a space 4 ft. in diameter at the center is not available; deducting this area leaves a working surface of about 65 sq. ft. Estimating the size of the mineral particles to be 0.005 in., there will be  $\frac{1}{2400}$  cu. ft. of mineral for each square foot of surface, or  $\frac{65}{2400}$  cu. ft. for the entire working surface. The table, revolving approximately once every minute or 1,440 revolutions in 24 hours, gives  $\frac{93600}{2400}$  or 39 cu. ft. of mineral in 24 hours, which corresponds to the passage of 390 cu. ft. of ore passing over the table, which, assuming 13 cu. ft. to the ton, gives 30 tons of material treated. This estimate is on the assumption that all conditions are perfect and is given solely to show the basis of the claim for a large capacity. In one case, with an ore carrying about 8 per cent. of lead or 9 per cent. of lead sulphide and passing a 200-mesh screen, a capacity of 22 tons in 24



hours was developed, with a 60 per cent. saving of the lead-content.

These theoretical figures being based on perfect conditions can hardly be considered as possible in practical work, but actual practice has shown that from 30 to 50 per cent. of the theoretical capacity can be relied upon.

Another point, not dwelt upon in my former paper, is that the table was not affected by a variable feed, provided that the feed did not exceed the point of covering the surface with more than one layer of mineral. This fact has been proven both practically and theoretically. In practice it was shown by feeding a small quantity of pulp and arranging the adjustments to approximate closely those best to be used in the determinative test. In theory the layer of mineral should not exceed one particle in depth, and assuming that all adjustments are arranged for the proper working of the full capacity, it naturally follows that the mineral particles will not be disturbed by the washing action of the water. If the feed be cut off to even a small fraction of the full load, the mineral particles will settle to the table just the same as before, but will cover only a proportionately small area of the surface. The silica will be acted on in a similar manner, and the metallics will not be influenced any more than if the surface were entirely covered. This question can be summed up in the statement that the table does not need a bed before it will work normally; in other words, each particle is practically treated by itself.

Referring to the plan of the top (Fig. 3), I would call attention to the card of the motions. The combination of the rotating and the reverse vanning motions give a peculiar "panning" action, and by placing a plaque on the top containing a small quantity of pulp and water, a perfect stratification will take place in traversing a distance of 3 or 4 feet. In this manner the size and speed of the vanning motion is sometimes tested.

In some cases the addition of auxiliary sprays was found to be of advantage, as is the case in separating minerals.

### The Metallurgy of the Homestake Ore.

Discussion of the Paper of Mr. Charles W. Merrill, p. 585.

MARK R. LAMB, Torres, Sonora, Mexico (communication to the Secretary\*): I have read Mr. Merrill's paper with much interest, and cannot but accept his challenge as regards the cost of the clean-up at the Homestake mill.

If I understand him correctly, the two men take the precipitates from the presses in about six hours, and do nothing towards refining them. If that is so, it would correspond to the work four men do here in about eight hours—that is, to clean the box and to place the precipitates in the driers.

In order to make a fair comparison, the bullion in each case should be stated in ounces, since it is as difficult to precipitate silver as gold, and our bullion is mostly silver. Bullion to the value of \$50,000 is equivalent to a weight of about 2,500 oz. At our last clean-up eight standard bars, each 1,100 oz. in weight, were made, which equals a total weight of 8,800 oz. Four men had the precipitates drying by 3 P.M.

This plant, the Minas Prietas works of Charles Butters & Co., at Minas Prietas, has square-bottomed electrolytic vats, an arrangement which necessitates the handling of all the precipitates in buckets. The Butters plant at Virginia City, Nev., is equipped with conical vats, and probably could make a still better showing under similar conditions.

Another point is that common labor only is required. (Ordinary laborer's wages at Minas Prietas are \$0.70 gold.) I may say also that, except for the convenience of the refinery, the box could be cleaned monthly, instead of bi-monthly, at only a slight additional expense.

The comparison, if made a few months ago, would have been much more in favor of the electrolytic process, as the tailings handled at that time contained a large quantity of copper, and the amount of zinc-dust required would have been enormous.

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\* Received October 17, 1903.

To resume: 2,500 oz. of bullion are handled by 2 men at Homestake, and 8,800 oz. by 4 men at Minas Prietas.

MR. MERRILL (communication to the Secretary): Mr. Lamb's contribution to the data of clean-up costs under various methods of precipitation from cyanide solutions is interesting and welcome, and I trust will be followed by some one who is handling large quantities of bullion by the zinc-shavings method. At one of the largest plants in New Zealand, handling 25,000 tons per month, I am informed that 7 men are kept busy constantly in cleaning up the zinc-boxes.

Our last clean-up at the Homestake Plant No. 1 consisted of approximately 3,233 oz. of precious metals, which was removed from the presses by 2 men in 5 hours, a duty of 323 oz. per man-hour. Mr. Lamb's men, in removing electrolytic slimes, handled 275 oz. per man-hour.

I do not know how the presence of silver affects the percentage of precious metal in electrolytic slimes; but in zinc-dust precipitation it greatly increases the proportion of precious metal. In the case of the Montana Company's plant at Marysville, Montana, I recollect that, in one instance, when we were running on comparatively high-grade silver tailings, our zinc-dust precipitate from the strong solution contained nearly 40 per cent. of precious metal.

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## A Laboratory Study of the Stages in the Refining of Copper.

Discussion of the Paper of Messrs. Hofman, Green and Yerxa, p. 671.

(Atlantic City Meeting, February, 1904.)

LAWRENCE ADDICKS, Perth Amboy, N. J. (communication to the Secretary\*): The excellent series of photo-micrographs given in this paper shows clearly the relation between micro-structure and cuprous oxide in refined copper. The authors suggest that this method of examination might be substituted for the "button" method now used to control the refining of cop-

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\* Received February 23, 1904.

per and that, with the aid of a planimeter, it might displace the present tedious methods of analysis for oxygen-content.

The photographs of the button-fractures which accompany the other data bear witness, however, to the fact that the fracture at various stages is quite as characteristic as the micro-structure—which is to be expected, as one depends upon the other—and to a practiced eye enables the process to be followed as closely as would be possible with micro-samples except toward the very end of the poling operation. But at that stage we have the surface or set of full-size castings to guide us. I should like to emphasize the fact that it is not a certain percentage of cuprous oxide in the casting which is sought, but such a quantity as will give a mechanically perfect product. This quantity varies through a very small range depending upon the size and shape of the casting, as the authors point out. The pitch is easily brought close to the proper point with the guidance of the buttons, and any final adjustment made by judging the surface of a larger casting. Personally, I cannot see that the use of micro-samples would give any better guidance, and with anode copper in which are varying quantities of numerous impurities, the micro-structure would not be so readily interpreted.

As to the tediousness of the analysis for oxygen, the simplest way to avoid this is to omit it altogether. The buyer of refined copper wants two things;—satisfactory electrical conductivity and a mechanically perfect casting. The conductivity is specified because so much copper finds its way into electrical uses. The purchaser relies on the conductivity requirement to give him a copper so pure that it has satisfactory physical properties, and any analysis of refined copper is rarely necessary. Small variations in pitch are noticeable in the set of the castings long before the conductivity is appreciably affected.

In regard to the development of the structure of samples of copper, I have had good results by treating the specimens for about 30 seconds in boiling aqua regia diluted with an equal volume of water. Under this condition comparatively little preliminary polishing is required.

H. O. HOFMAN, Boston, Mass.\*:—The idea that prompted us to suggest the use of the microscope for the control of the end

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\* Communication received by the Secretary, May 18, 1904.

of a refining operation was that we had found that the point to which the reduction of cuprous oxide was carried with wire-bar copper varied at different plants. True, the final adjustment of the metal in the furnace is made either alone by the condition of the surface of a larger casting, or it is supplemented by mechanical tests, or by the brass test. A test for conductivity always follows, if the metal is to be used for electrical purposes. These tests give the all-important practical answer to the suitability of the copper for certain purposes, but they do not give the reason for this fact. The microscope will do it, if a correct percentage of cuprous oxide is the sole cause; it may do it, if with the cuprous oxide there are other influences that cause a failure in one case and a success in another. Investigations to corroborate or refute the second view can, however, be carried out only at the refinery itself, where furnace-experience and laboratory-method go hand in hand.

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### Investigations in Thermal Chemistry Showing Atomic Heat-Valency.

A Discussion of the paper of Mr. H. P. Gillette, p. 702.

ALFRED H. COWLES, Cleveland, Ohio (communication to the Secretary\*) :—Mr. Gillette's paper and his deductions seem to me of the very greatest importance, if the validity of his conclusions and figures can be confirmed. His work seems to solve an unfinished problem that I attempted to solve on the same lines about two years ago, at a time when I was preparing a paper<sup>1</sup> on another subject for the American Electrochemical Society. The following paragraph is quoted from the paper referred to :

"I feel convinced that a further study of the heats of combination now known for so many compounds, which merely express in other terms their voltages of disruption, will show us the allowance that must be made as a departure from unity in the case of each element, for the intrinsic charge of its elementary molecule, and thereby render possible the creation of a table of apparent relative atomic capacities for electricity, similar to the table of atomic weights. With the numerals of this table known, thermal reaction or energy-transformations will be read with the same ease that we now read and trace the shifting of mass

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\* Received March 25, 1904.

<sup>1</sup> *Proceedings of the American Electrochemical Society*, vol. ii., p. 212.

in chemical reactions. *The reciprocal of the voltage of disruption is the capacity of the molecule at unit potential for electric charge.* A table of these reciprocals would be analogous to an obverse side of a table of molecular weights. The data for such a table are already extant, and the writer ventures the prediction that the rest will soon be solved."

In the above statement that "the reciprocal of the voltage of disruption of a compound molecule is the capacity of the molecule for electric charge at unit potential," Hemholtz's temperature-factor should be embodied, to make the statement completely true.

Voltages of disruption of compounds in electrolysis measure energy, and are directly proportional to heats of combination, when allowance is made for the combining valency. It is as though the calorie were a volt of different magnitude; hence, while I was considering the problem from the electrical side, the data involved and the conclusion to be arrived at were really the same. If Mr. Gillette has solved this problem, it seems as though his work would elevate chemistry to the dignity of an exact science.

H. P. GILLETTE, New York City (communication to the Secretary\*):—Since the preparation of my paper, I have discovered what appears to be a relation between the atomic heat of solution and the specific heat of certain elements. In studying "TABLE III.—*Heats of Solution*," I observed that the metallic elements having a high valency for their heat of solution, have a low atomic weight: a fact which led me to compile Table IV.

TABLE IV.—*Relation Between Atomic Weight and Atomic Heat of Solution.*

Element.	Atomic Weight.		Atomic Heat of Solution.		Product.
Mg.....	24	x	11.5m	=	276m
Ca.....	40	x	7 m	=	280m
Ba.....	137	x	2 m	=	274m
Mn.....	55	x	5 m	=	275m
Cu.....	63	x	4.5m	=	283m
Cd.....	112	x	2.5m	=	280m
Sr.....	86	x	4.5m	=	387m
Zn.....	65	x	6 m	=	390m
Co.....	59	x	6.7m	=	395m
Pb.....	206	x	2.7m	=	556m

\* Received April 25, 1904.

In Table IV., as in Table III.,  $m = 3.2$  cal., and the values in the third column of Table IV. have been taken as rough averages of those given in Table III. It will be noted that the product of the atomic weight by the atomic heat of solution is approximately  $280m$  for six metals;  $1.4 \times 280m$  for three elements; and  $2 \times 280m$  for one metal.

The fact that the atomic weight multiplied by the atomic heat of solution, at least in certain cases, appears to be a constant, points toward a definite relation between atomic heat of solution and specific heat; for it is well known that the product of an atomic weight of an element multiplied by its specific heat is approximately 6.3.

In order to show that the ratio of the atomic heat of solution to the specific heat is approximately constant, I have prepared TABLE V.

TABLE V.—*Ratio of Atomic Heat of Solution and Specific Heat.*

Element.	Atomic Heat of Solution.			Specific Heat.		
Mg.....	11.5m	÷	0.25	=	46m	
Ca.....	7 m	÷	0.17	=	41m	
Ba.....	2 m	÷	0.047	=	42m	
Mn.....	5 m	÷	0.12	=	42m	
Cu.....	4.5m	÷	0.094	=	48m	
Cd.....	2.5m	÷	0.057	=	44m	

While I realize that these tables show only in a general way that atomic heat of solution is a function of specific heat, and that much yet remains before any exact law of relationship is proved, still it has seemed to me worthy of record that, roughly speaking, the atomic heat of solution of certain metallic elements bears a constant ratio to the specific heat. I hope, in a subsequent paper, to present what appears to me to be the philosophical explanation of these facts.

## The Copper-Deposits of the Kaibab Plateau, Arizona.

A Discussion of the Paper of Mr. E. P. Jennings, p. 839.

HORACE F. LUNT, York, Montana\*:—Two years ago I visited the region described by Mr. Jennings and assisted in the examination of a somewhat similar group of deposits in the White Mesa sandstone on the Navajo Indian Reservation, 125 miles north of Flagstaff and 50 miles east of the Kaibab plateau.

There, the Triassic red sandstone is less than 200 ft. thick, and is overlain by a stratum from 200 to 250 ft. thick of white sandstone, which gives the name of "White Mesa" to that locality. The white sandstone shows all phases of cross-bedding in great complexity, and evidently marks a period of oscillation and subsidence of the water, or an elevation of the land, at the close of a geological epoch.

The prevailing color is white or gray, but occasionally it is red. In structure, it is composed of well-worn grains of quartz-sand, from which most of the cementing material has been leached out, leaving it soft and friable. Possibly the cement was ferruginous, as is indicated by occasional traces of red color.

The copper-deposits consist of a replacement of the cement of one particular cross-bedded stratum of the sandstone by chrysocolla, having also at times various quantities of tenorite in grains or masses. Some specimens of the ore contain as much as 32 per cent. of copper. Generally the copper-deposits are a few hundred cubic feet in size, and no strict line of demarcation is shown between them and the surrounding sandstone, the bluish-green color of the chrysocolla gradually fading into the white of the sandstone.

The ore is apparently associated with small vertical crevices or fissures. Some of the deposits show a distinct vein-structure, while others do not. The origin of the ore is not very clear. Undoubtedly the solutions, coursing through the crevices noted above, played a prominent part in its deposition, as possibly

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\* Communication received by the Secretary. June 7. 1904.



these solutions carried and deposited the copper. On the other hand, it may have been leached from the sandstone and precipitated by the solutions. In the latter case it is not unlikely that part of the copper, escaping precipitation, was carried downward and deposited in the Carboniferous limestone.

These deposits have never been worked commercially, on account of the scarcity of water in the neighborhood. I understand that similar deposits in Utah are mined at a profit.

## ERRATA.

Page xxviii—W. F. Stevens is erroneously reported deceased.

Page 109—Title of Fig. 1, "Naticoke" should be "Nanticoke."

Page 902—Footnote reference <sup>15</sup>, add the date Feb. 19, 1903.

Page 903—Footnote reference <sup>17</sup>, add the date Feb. 19, 1903.

Page 906—Bottom of Table II., add

Ounces fine gold = . . . . .	3352.6	
Ounces fine silver = . . . . .	1400.	
Total ounces doré bullion = . . . . .	4752.6	
Cost of refining per ounce doré bullion, . . . . .		\$0.0984

Page 907—Bottom of Table III., replace

Total oz. bullion, . . . . .	1567.6	
Cost per oz. bullion, . . . . .		\$0.221
by		
Ounces fine gold = . . . . .	914.	
Ounces fine silver = . . . . .	680.	
Total ounces doré bullion = . . . . .	1594.	
Cost of refining per ounce doré bullion, . . . . .		\$0.217

Page 917—Line 26, "iron-ore and silica" should be "suitable fluxes."

Page 917—Line 27, after the words "from which the lead separates," add "This slag would have to be re-run in a shaft-furnace in order to recover lead, and small values of gold and silver."

Page 959—In table, top of page, under column 4, [arsenic], 0.011 should be 0.110.



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